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MINING, TREATMENT METHODS AND COSTS, MENANTICO SAND AND GRAVEL CO., MILLVILLE, N. J.

BY

HUGH HADDOW, JR.

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MINING, TREATMENT METHODS AND COSTS, MENANTICO SAND AND GRAVEL CO., MILLVILLE, N. J.[1]

By Hugh Haddow, jr.[2]

INTRODUCTION

This paper describing the methods of recovery and treatment of a sand and gravel deposit and the preparation of these materials for a number of special markets is one of a series being prepared by the U. S. Bureau of Mines. It is of particular interest to those sand and gravel operators who have a surplus of fine sand which must be sent to waste, both from the point of view of equipment and the variety of markets supplied, and in that it describes one of the few sand and gravel plants which have resorted to hydraulic classification for sizing fine sand.

ACKNOWLEDGMENTS

The author wishes to acknowledge the assistance of J. R. Thoenen, mining engineer of the Bureau of Mines, in the preparation of this paper.

HISTORY

In the early operation of the deposit sand and gravel for making concrete were the only products. Later on the demand for special sands necessitated separate plant units for their preparation, which was followed by the erection of another separate plant for the utilization of the fine sand wasted from former operations, as markets were developed for this material. The present plant is the result of a gradual development brought about by the necessity of producing various special kinds or gradings of sands and gravels.

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6420."

2 One of the consulting engineers of the Bureau of Mines; vice president and general manager, Menantico Sand and Gravel Co.

Previous to the formation of the present company the deposit had been worked in a small way for the production of concrete gravel and blast sand. In the spring of 1914 the present company took over the property and constructed an entirely new plant. At that time the only outlet for material was a small market for concrete gravel in Atlantic City and a fair market for blast sand in New Jersey and Eastern Pennsylvania. Several sand plants were already in operation in this section but they were almost entirely occupied in supplying sand to glass factories, foundries, and for water filtration.

The property consists of about 300 acres with a small stream, Menantico Creek, running through it close to the western boundary. The Pennsylvania Railroad tracks traverse the property on the southern edge so that excellent transportation is afforded.

DESCRIPTION OF DEPOSITS

The deposits at Millville are all of fine sand and gravel that offer no serious difficulty to the operation of a pump dredge. On the property are two classes of deposits which are called Highland and Lowland. The Highland deposit extends from 10 to 20 feet above water to an average of 20 feet below water level. This deposit has no overburden and supports a weak growth of scrub oak and pine. The Lowland deposit is adjacent to the stream and extends from 2 to 5 feet above water to an average depth of 22 feet below water level. The overburden on this deposit is decayed vegetable matter mixed with sand and forms a black mat from 6 inches to 3 feet in thickness over the deposit.

PHYSICAL CHARACTERISTICS

Gravel deposits in all of Southern New Jersey are of small area and adjacent to small streams. The gravel itself is unusually small with almost none over 2 inches in size. On the other hand, sand of almost every size is found in abundance.

The Lowland deposit is comparatively shallow, but consists largely of gravel and coarse sand, and was the first deposit worked because it contained from 20 to 25 per cent of gravel and about 35 per cent of coarse sand which were the only products of value during the early operation of the deposit. The balance of fine sand was then considered as waste.

Chemically, the material shows an analysis of from 96 to 98 per cent silica and is uniform throughout. The sand is angular and hard, and the gravel is practically all quartz.

PROSPECTING AND EXPLORATION

The first explorations were by means of test pits dug by hand. These proved of little value as they could only be sunk to a limited depth. A contract was then made for drilling test holes over that portion of the deposit which it was desired to explore. These test holes were put down by hand at a contract price of $1.50 per foot and spaced approximately 100 feet apart in each direction.

At present the method of prospecting is to dig test holes with a small Hayward clam-shell bucket worked by hand. The bucket digs a hole about 12 inches in diameter and is used above the water line inside a steel-pipe casing. Below the water line a 4-inch pipe is sunk as a casing within which a drill tool is operated. This tool is made from a piece of 3-inch pipe about 4 feet long with one end filed off to a rough cutting edge. A few inches above this edge a flap valve is placed opening upward. The tool is suspended from a light tripod by a half-inch rope running through a pulley and is operated by hand. Three men are used in this work and average about 50 feet of drilling per day at a cost of 30 cents per linear foot.

CHOICE OF METHOD

Originally the desired product was concrete gravel and coarse sand which occurred in the best quantities in the Lowland deposit. The overburden being of no value whatever, had to be disposed of as quickly and economically as possible, so it was cast into the pond formed by previous workings.

In order to use the equipment taken over in the purchase of the property, consisting of a Browning locomotive crane and an "A" frame derrick, both equipped with 1-yard buckets, a screening plant was constructed close to the railroad track and dry plant. The material was dug by the crane and derrick, loaded in 4-cubic yard side-dump cars and hauled by a steam locomotive to the screening plant.

The cars were dumped into a hopper and the material raised by a continuous bucket elevator to a jacket screen which separated the sand from the gravel. The gravel was loaded by gravity either into cars or a storage bin, and the sand was discharged into a wash box which separated it into coarse and fine sand. The coarse sand was for use as blast sand and the fine sand was wasted.

The disposal of this fine sand eventually became an acute problem so some change was necessary. Then too, by this time, a change in the market was evident. An increased demand for a properly prepared concrete sand was evident, as well as a demand for a fine sand that could be used for plastering and brick mortar. In order to meet this demand it was decided to entirely revamp the plant.

Due to the shallowness of the deposit and because it contained no boulders or large stones, it was decided that a pump dredge would handle the material cheaper than any other device. In order to avoid pumping long distances, an inexpensive screening plant was built which could be replaced by another plant in a new location when the pumping distance became too great. As steam power only was available, the dredge was constructed to accommodate boilers and an engine, and as there was no way of getting a dredge hull to the property it had to be built at the site.

In 1926 electric power became available and since the boiler equipment was reaching a state where renewals were necessary it was decided to equip the plant throughout with electric machinery. This change was made during 1926 and the electric machinery has proven more efficient and less expensive to operate than the steam-operated equipment.

During all this time the market was broadening and a demand was appearing for a greater variety of grades in both sand and gravel.

The demands were met and in some cases anticipated by the construction of new plants for special purposes, until at present four plants are in operation consisting of a wet-screening plant for the preparation of concrete gravel, concrete sand and fine or plastering sand; a dry plant for the production of blast sand; a re-screening plant for the preparation of filter gravel; and an hydraulic-classifier plant for the preparation of special sands for such purposes as molding, core, filter, and slate-rubbing sand.

This last plant was built to utilize the fine sand which had been discarded as waste during previous operations. The flow sheets of the various plants are shown in figures 1, 2, 3, and 4.

DREDGE

When originally built, the dredge hull had to accommodate a Corliss engine and boilers so that it is somewhat larger than is necessary at present. The hull is 30 by 70 by 5 feet deep and is built throughout of long-leaf yellow pine. The deck and sides are of 3-inch planks and the bottom planks are 4 inches thick. A deck house of galvanized iron covers all equipment. The details of this hull are shown on the accompanying plan (fig. 5).

The equipment consists of a 12-inch Morris Machine Co. sand pump with Taylor-Wharton manganese-steel removable liner, side plates, and impeller. The pump is driven by a direct-connected 300-hp. General Electric synchronous motor using 400-volt, 3-phase, 60-cycle, current. Synchronous motors were selected on account of low initial cost and a power factor affording a 5 per cent reduction of current consumption.

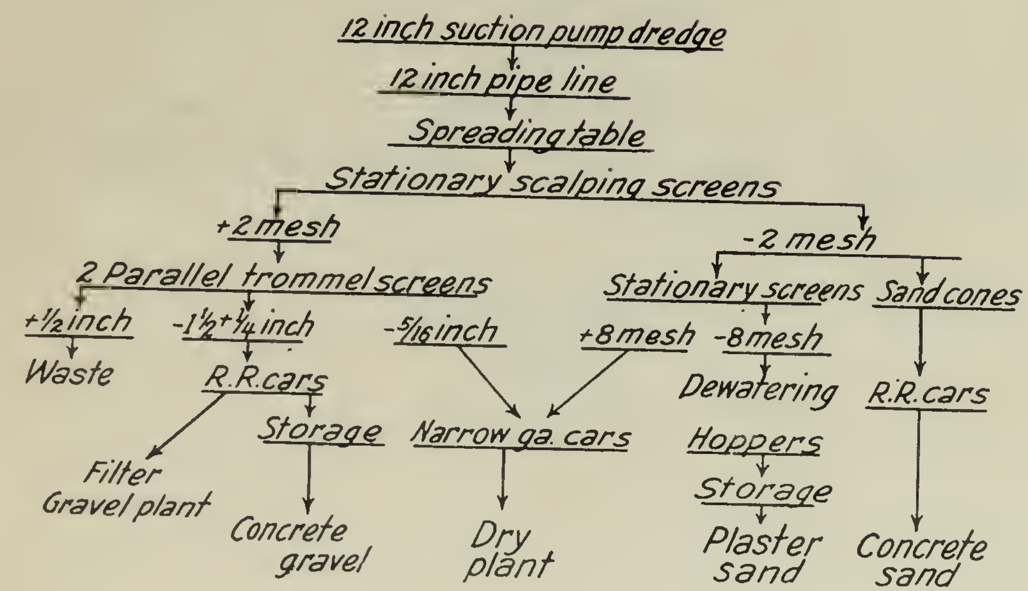


Figure 1.-Flow sheet of sand and gravel or wet plant

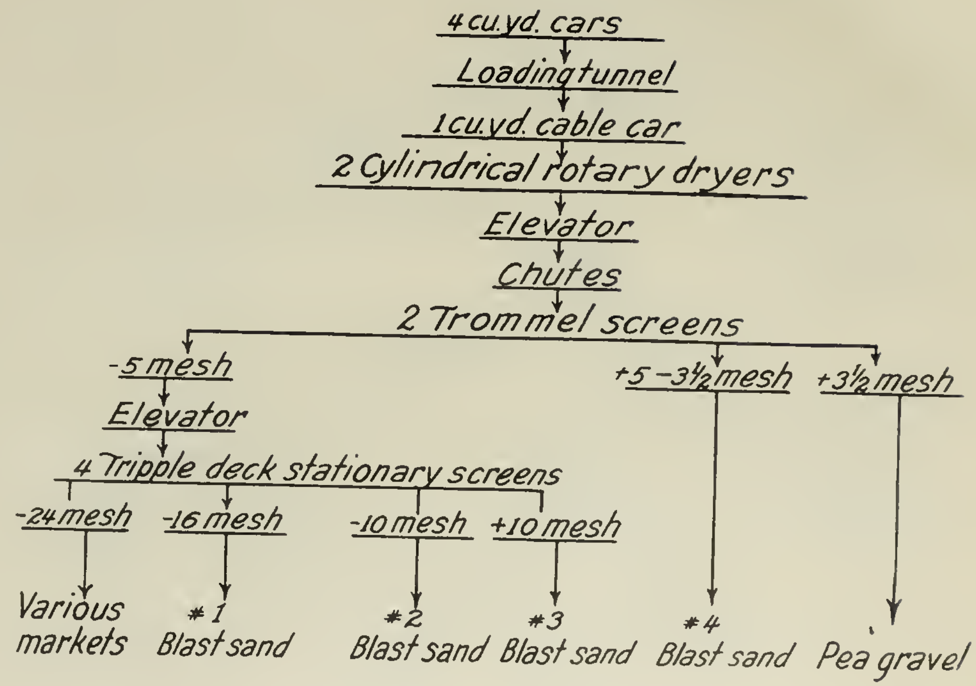


Figure 2.-Flow sheet of blast-sand or dry plant

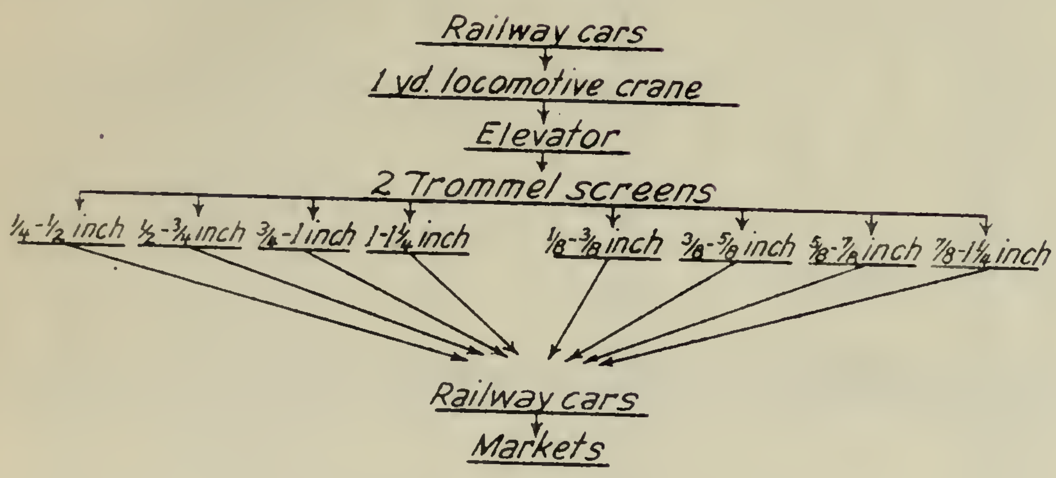


Figure 3.-Flow sheet of filter-gravel plant

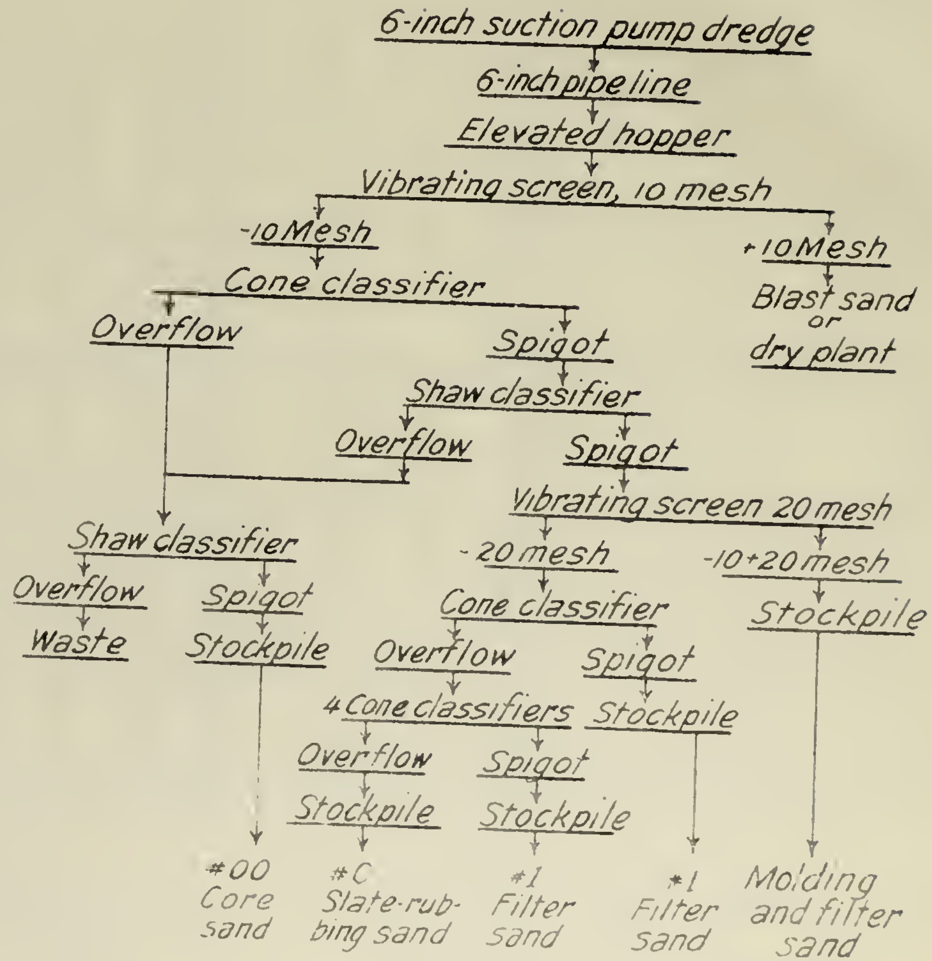


Figure 4.-Flow sheet of classifier plant

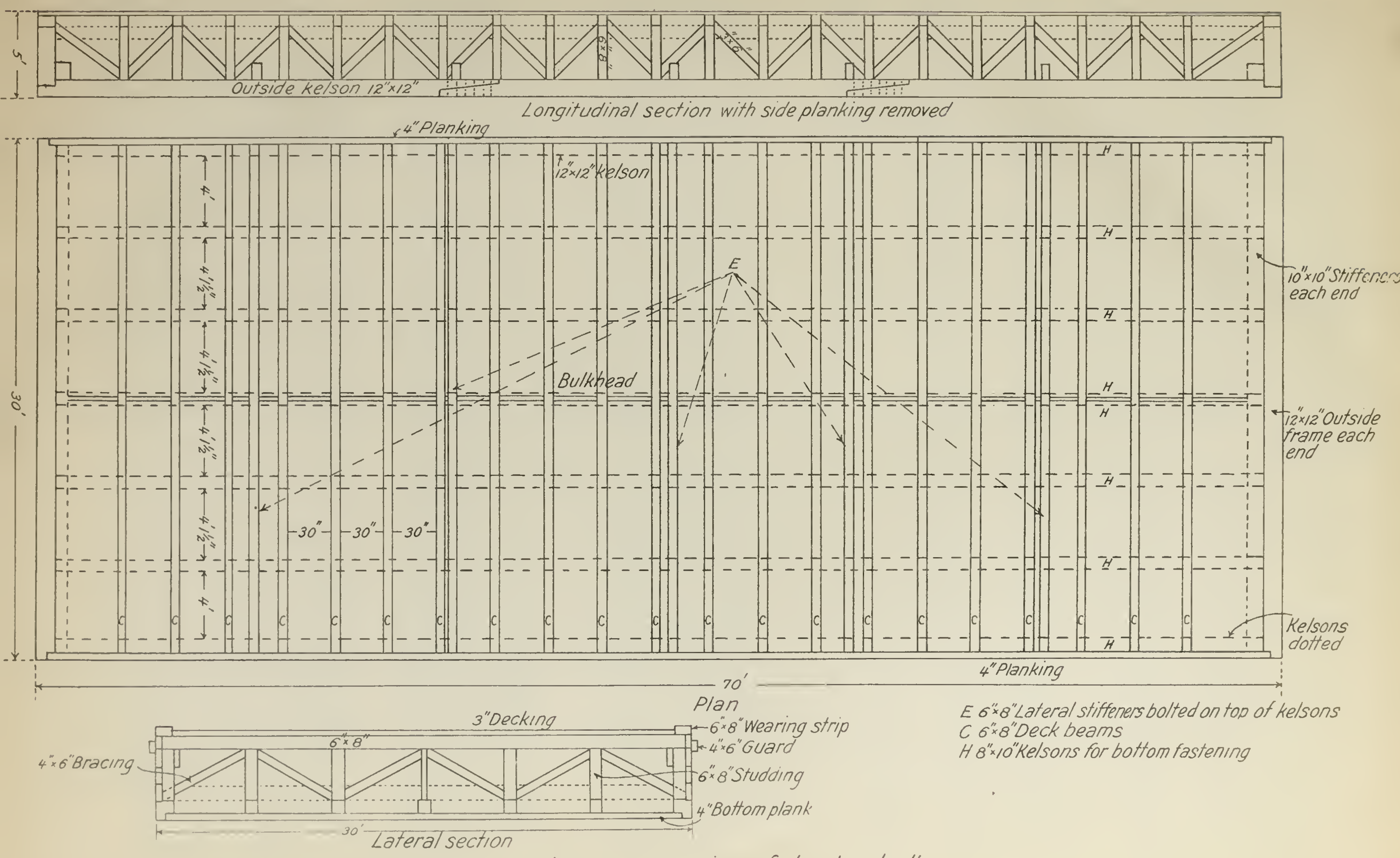


Figure 5:-Framing of dredge hull

To eliminate starting troubles the motor is equipped with an automatic starter and push-button control so that to stop or start the pump all the operator has to do is to push the button.

For priming there is a $1\frac{1}{2}$ inch Nash vacuum pump driven by a 2-hp. motor and there is also a 2-inch D.S.V.M. pump driven by a $7\frac{1}{2}$-hp. motor which provides a water seal to keep sand out of the main bearing.

The suction line is suspended from an "A" frame by wire cables and is raised and lowered by a hoisting engine driven by a 20-hp. induction motor.

Current for these small motors and for lighting the boat is furnished from a bank of three 10 kv.a. transformers which reduce the 4,000 volts to 220 volts for the motors and 110 volts for the lighting circuit.

The dredge is moored to convenient points on the shore by wire cables which are operated by hand winches.

PIPE LINE

Various types of pipe have been used in the discharge lines, but the most satisfactory has been pipe made by the American Rolling Mills Co. of a special analysis steel with welded seams. This pipe is carried over the water on pontoons each 8 by 4 by $2\frac{1}{2}$ feet. For convenience in handling, the pipe line is generally made up with rubber sleeves about every 40 feet. The pipe line is 12 inches in diameter throughout and at the screening plant is carried up a long incline to a point of discharge 32 feet above the loading track.

SCREENING PLANT

The pipe line discharges on the narrow end of a fan-shaped spreading table which slopes 7/8 inch in 1 foot toward the stationary screens. This table is 20 feet wide at the lower end and spreads the material in a thin layer. The material flows from this table onto a woven wire screen 20 feet long and 4 feet wide. This screen is set at an inclination of about 45° and so located that the discharge from the table hits the top of the screen.

As there is practically no material over 2 inches in size the screen used is 2 meshes to the inch of 0.162-inch wire, giving a clear opening of 0.338 inch. The purpose of this screen is to separate the gravel from the sand; the gravel is discharged into a chute which conveys it to 2 parallel revolving screens each 16 feet long and 36 inches in diameter and equipped with sand jackets 12 feet long. The purpose of these screens is to remove small roots or trash from the gravel and also to screen out any sand not removed by the stationary screen. The inner section of the screen is punched metal with holes $1\frac{1}{2}$ inches in diameter, while the sand jacket is punched with holes 5/16 inch in diameter.

The material passing the inner screen and retained on the sand jacket is therefore gravel from 5/16 to $1\frac{1}{2}$ inches in diameter and is chuted directly into cars, there being no gravel bins at the plant. The sand passing through the jackets enters a hopper from which it is loaded by gravity into small cars which convey it to the dry plant over a narrow-gage track.

The sand passing the 2-mesh stationary screen is carried into a series of sand cones of the Dull type which dewater the sand and also remove any excess of fines which may be present. These cones discharge the dewatered sand directly into cars for shipment as concrete sand. When fine sand for plastering is desired another screen similar to the 2-mesh screen but with 8-mesh openings is placed directly ahead of the 2-mesh screen. This screen separates the sand into fine and coarse sand. The coarse sand rejected by this 8-mesh screen enters a hopper and is loaded by gravity into narrow-gage cars which take it to the dry plant. The fine sand passing the 8-mesh screen goes to the dewatering hoppers and is loaded into cars for shipment as plaster sand.

To facilitate handling these screens they are made in three sections, each 6 feet 8 inches long on wooden frames.

DRY PLANT

The dry plant is operated to produce blast sand and filter sand which are shipped in box cars. The raw material used is the coarse sand produced at the wet plant, which is loaded at the wet plant in 4-cubic-yard cars on a 3-foot gage track and hauled by gasoline locomotive to an elevated trestle and dumped on a storage pile. This storage pile covers a concrete tunnel in which there is a 24-inch gage track carrying two 1-cubic-yard Kopple side-dump cars.

The sand is loaded into these cars through gates in the roof of the tunnel and the cars are then hauled up an inclined trestle by cable to the receiving hopper of the dryers.

There are two rotary-type direct-heat dryers, each with a rated capacity of 20 tons per hour, and using low-volatile coal for supplying heat, the consumption of fuel being about $1\frac{1}{2}$ tons per day in each dryer. The dryers are belt driven by a 50-hp. synchronous motor.

The wet sand is dumped from the Kopple cars into feed hoppers located over the dryer furnaces and is fed through a gate into the receiving end of the dryers. The dryers make 15 r.p.m. and discharge the dry sand into a hopper. From this hopper the sand is picked up by a bucket elevator, raised 35 feet, and discharged into two parallel revolving screens, each 12 feet long and 36 inches in diameter. Each screen is in 2 sections, the first of which is equipped with wire cloth having 5 meshes to the inch and the second with wire cloth having $3\frac{1}{2}$ meshes to the inch.

The material passing the 5-mesh screen drops into a small hopper and is raised about 10 feet by a bucket elevator for further screening.

The material retained on the $3\frac{1}{2}$-mesh screen is oversize and is discharged into a loading bin for market as pea gravel. The material passing the $3\frac{1}{2}$-mesh screen also goes to a storage bin as No. 4 blast sand.

The sand which has passed the 5-mesh screen after being raised by the bucket elevator is discharged into chutes which open onto 4 parallel stationary screens. These 4 screens are alike and have triple decks 18 inches wide but varying in length. The upper deck, 20 feet long, is of 10-mesh wire cloth. Material retained on this screen is No. 3 blast sand and is chuted to a storage bin. The second deck of the screen is about 30 feet long and is of 16-mesh cloth. The material from this screen is also chuted to its bin ready for loading as #2 blast sand. The third deck of the screen is of 24-mesh cloth and the material retained on this screen is the smallest size or #1 blast sand. It also is discharged into a storage bin. All of these bins are inside the building so that the sand is kept dry.

The sand which passes through the last screen is too fine for blast sand, but is chuted into an inclosed bin and is sold as a special product for use in several different industries.

The gravity screens are assembled in sections, each 10 feet long and mounted on light wooden frames so that the replacements are easily made. The screens have an inclination of about 45°, so that the material flows over them slowly.

Six men are employed in this plant; a foreman who is also the locomotive crane operator, an engineer in charge of the motor room and who operates the hoisting engine that pulls the cars of raw material from the tunnel; a laborer who loads and discharges tunnel cars; a laborer who feeds the dryers and also attends the furnaces; a laborer in the screening plant and a laborer as helper on the crane. In loading cars the material is chuted into the hopper of a Pratt box-car loader and placed in the car; the loading is completed by the machine.

CLASSIFIER PLANT

During many years of operation a large amount of fine sand accumulated, which was looked upon as a waste product.

Chemical and physical analysis showed that this sand if properly graded could be made to meet the requirements of various industries.

As this sand was very fine, and of course wet, it seemed impossible to devise a screening method that would give satisfactory results. It was decided to attempt to get the required results by hydraulic classification. A plant was designed by and erected under the supervision of Mr. Edmund Shaw, using Shaw classifiers and cones. With a few minor changes and the addition of two Tyler vibrating screens this plant (see fig. 6) has proved entirely successful.

The flow sheet may appear somewhat complicated, but the plant has been able to produce several grades of sand that have found a ready market which is still being extended.

The classifier plant was erected 50 feet from the railroad track so that material produced could be stock-piled and loaded in cars by a crane as required. The raw material for this plant is the waste sand discharged into the old worked-out portion of the deposit. As the material is fine it can be readily pumped, and a 6-inch pump was selected for this purpose and mounted on a steel hull built at a local shop. The pump is belt driven by a 100-hp. synchronous motor and delivers the material through a 6-inch discharge line to a conical hopper at the top of the plant, 50 feet above the track elevation.

As the proper operation of this plant requires regular feed the discharge from the hopper is through a gate, the opening of which can be easily regulated. The hopper discharge is fed to a 10-mesh Tyler vibrating screen. All material retained on this screen is chuted to a stock pile and treated in the dry plant. The material passing the 10-mesh screen enters a cone classifier for preliminary classification (fig. 7). The spigot discharge from this cone enters a Shaw hindered-settling classifier (fig. 8) and is divided into two components. The overflow from this classifier unites with the overflow from the first cone classifier and all this material enters a second Shaw hindered-settling classifier. The spigot discharge from this second classifier is very fine sand and is stock-piled ready for loading as core or foundry sand. The overflow of this classifier is waste.

The spigot discharge from the first classifier falls on a Tyler 20-mesh vibrating screen. Material retained on this screen is therefore minus 10 plus 20 mesh and is stock-piled and used as filter and molding sand.

Material passing the 20-mesh screen flows into a third Shaw classifier, producing a spigot product of minus 20 plus 35 mesh, which is sold as filter sand.

The overflow from this classifier passes through 4 Deister-cone baffle-classifiers arranged in series. These reclaim any filter sand carried over in the overflow of the third Shaw classifier. The spigot discharge of these classifiers is therefore filter sand, while the overflow is a somewhat finer sand which is used in slate cutting and rubbing as well as for foundry work.

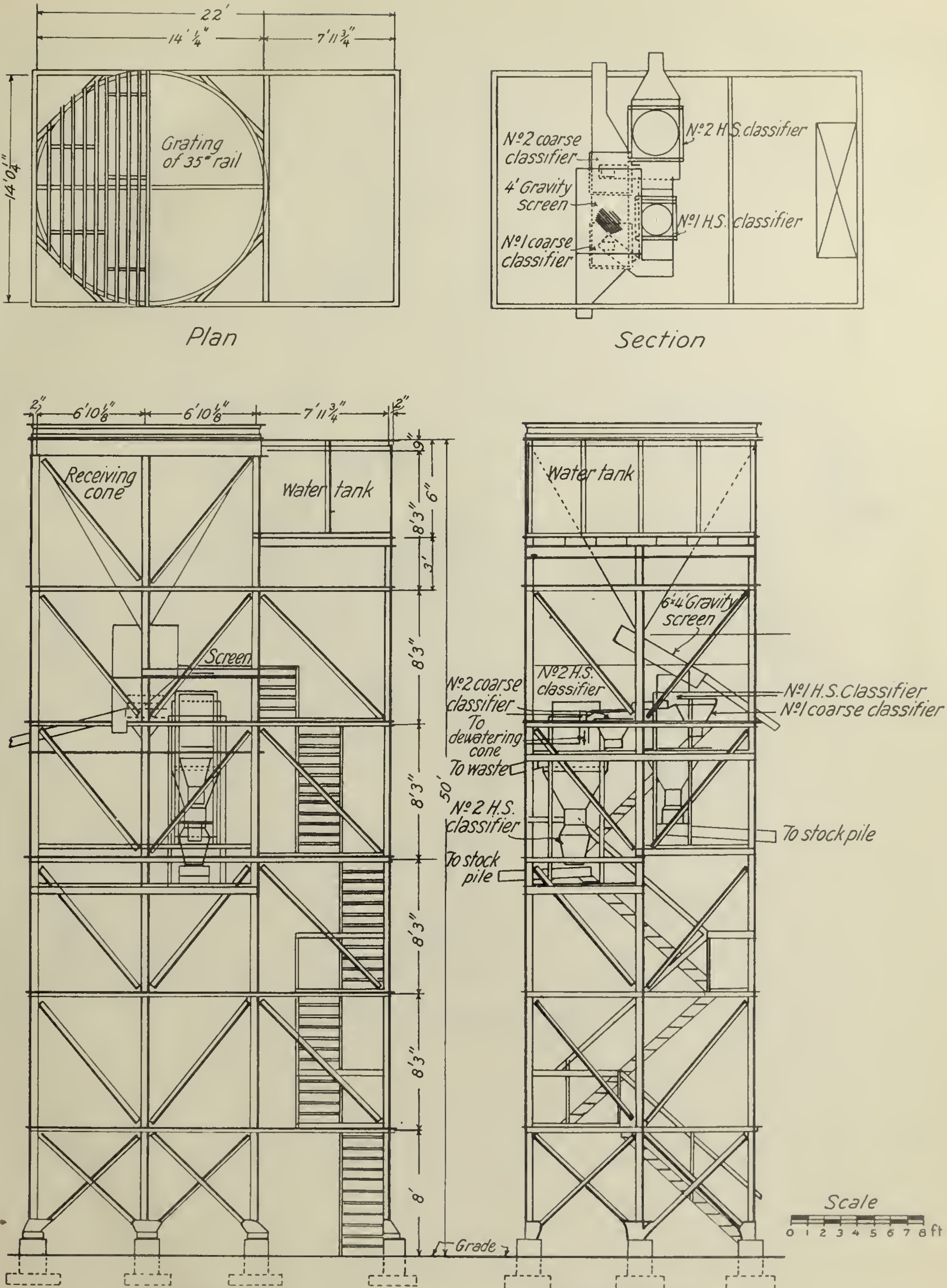


Figure 6.- Classifier plant

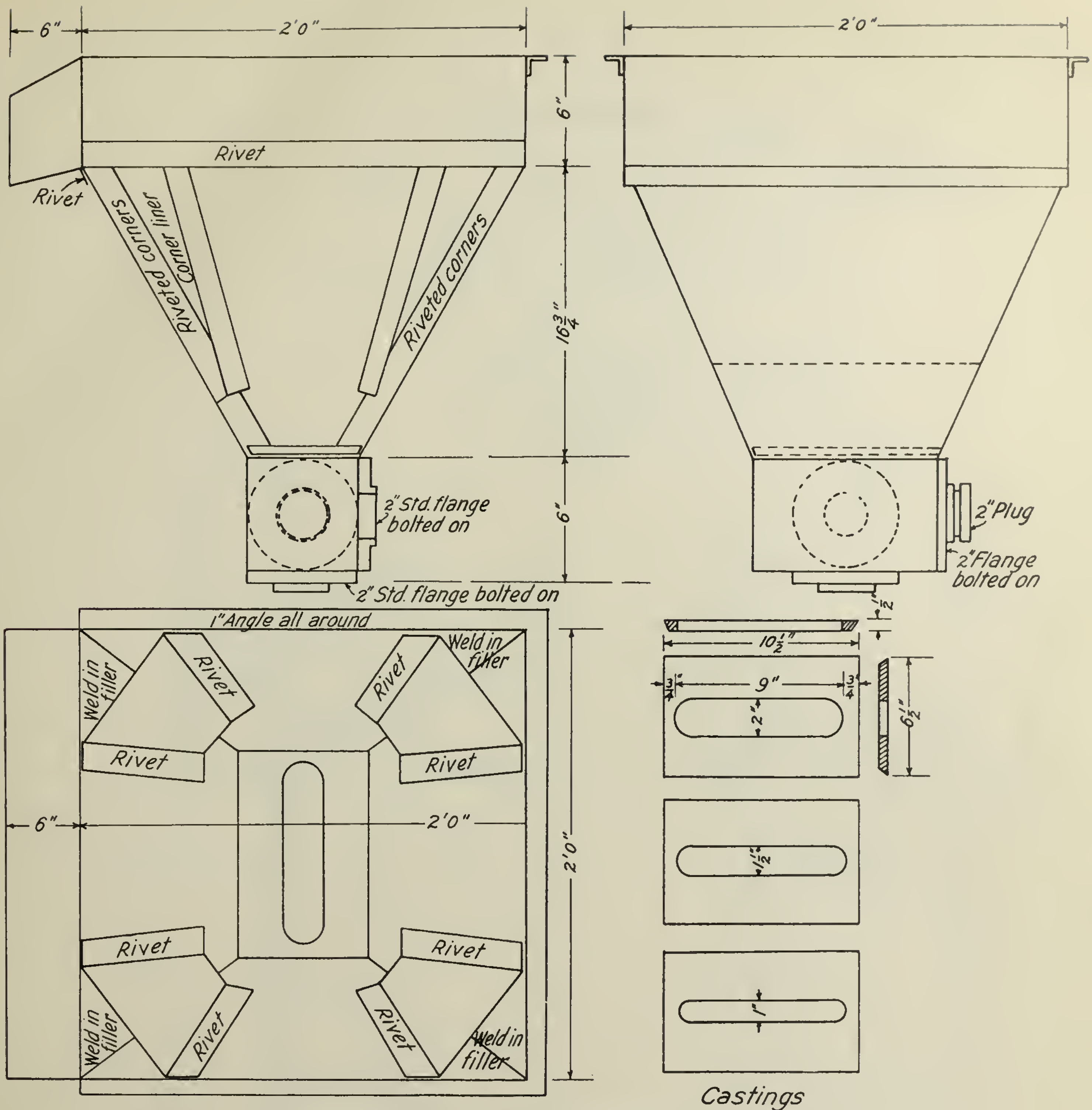


Figure 7.- Coarse classifier

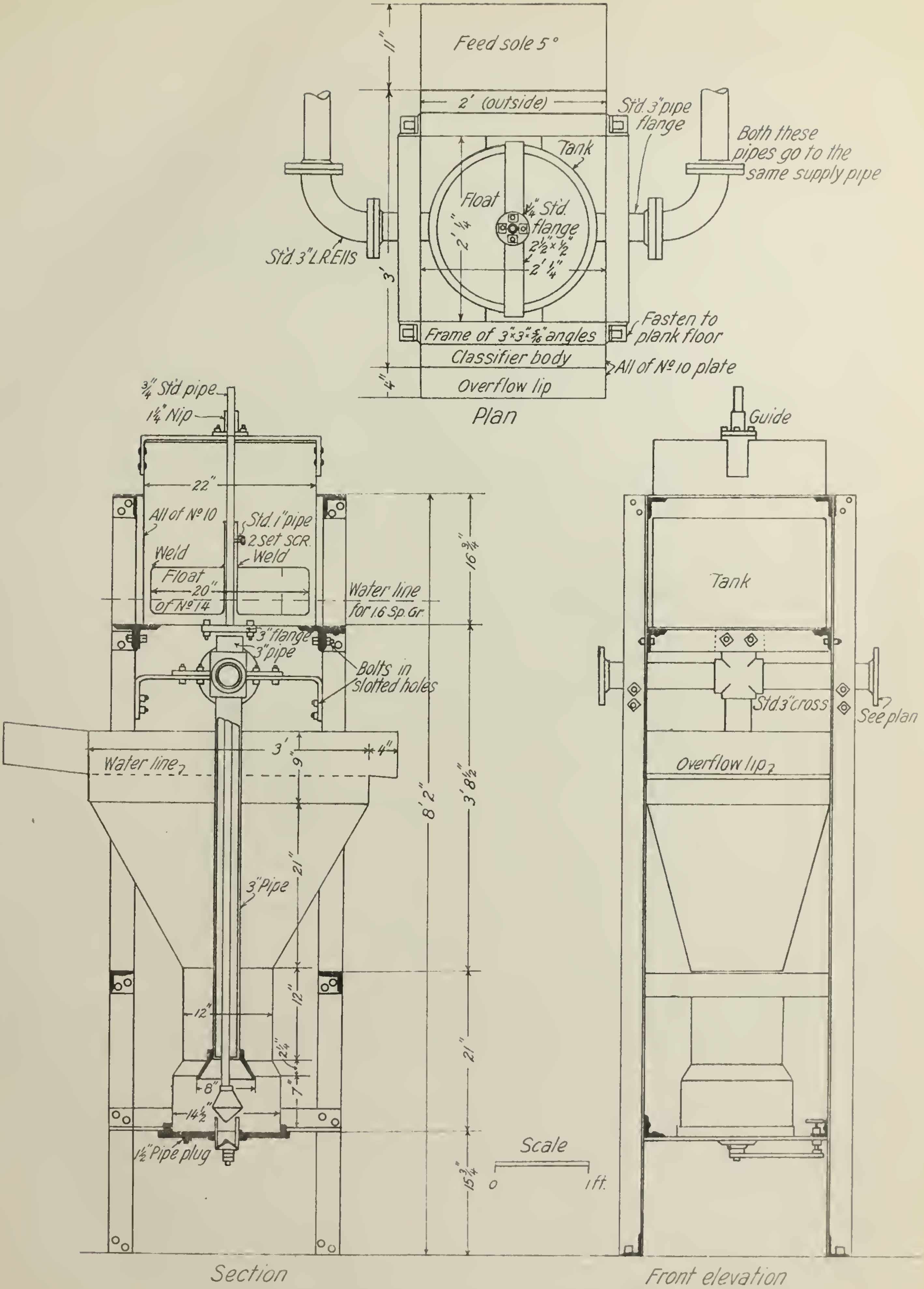


Figure 8.-Hindered-settling classifier

FILTER-GRAVEL PLANT

In order to produce the various sizes of gravel demanded for filters, a small gravel rescreening plant was built. This plant consists of two parallel revolving screens so constructed that the punched metal plates can be easily changed to accommodate the requirements as to size of gravel. The plant is 30 feet high, the gravel being raised by bucket elevator to the screens, discharged by gravity to ground storage and loaded into cars by a locomotive crane.

Water required in both the classifier and filter-gravel plants is supplied by a 6-inch centrifugal pump driven by a 20-hp. General Electric motor and in order to secure a constant head in the classifier plant a tank is provided at the top of the plant to which the water is pumped. Each classifier receives water by separate pipe line from this tank.

WAGE SCALE

		Per month
Foreman	2	$200.00
Watchman	1	125.00
		Per hour
Engineer	1	$ 0.675
Mechanic	1	.60
Pump Runner	1	.60
Crane operators	2	.55
Pump Runner	1	.55
Engineer	1	.55
Mechanic	1	.55
Loco. Engrs.	2	.50
Laborers	5	.45
Laborers	6 - 12	.40

Normally the working day consists of 9½ hours, five days per week, with 5½ hours on Saturdays, work being carried on continuously throughout the year.

SUMMARY OF COSTS

Period covered: 1927

Tons, sand and gravel produced 210,000

	Labor	Superintendence	Power	Fuel	Other supplies	Total
Operating costs						
Stripping by A - Frame derrick, casting overburden into pond	$0.008	$0.002		$0.001	$0.001	$0.012
Pumping and conveying	.031	.010	$0.074		.016	.131
Wet screening and loading	.089	.020	.003		.037	.149
Storage	.005			.003	.002	.010
Repairs and maintenance	.023				.004	.027
Miscellaneous expense	.013				.004	.017
Total operating cost of producing wet sand	.169	.032	.077	.004	.064	.346
Overhead						
Depreciation						.08
Depletion						.013
Taxes						.01
Insurance						.017
Selling						.04
Miscellaneous overhead						.01
Total overhead						.17
Total costs per ton						.516

DRY PLANT

SUMMARY OF COSTS

Period covered: 1927

Tons produced - 15,500

Operating costs per ton of dry sand

	Labor	Super-intendence	Power	Fuel	Other supplies	Total
Hauling	$0.037	$0.005			$0.062	$0.104
Drying	.226	.034	$0.130	$0.160	.112	.662
Screening	.158	.026			.054	.238
Loading	.057	.005	.010	.035	.027	.134
Miscellaneous plant ..	.020			.036		.056
Total operating cost dry plant	0.498	.070	.140	.231	.255	1.194
Operating cost of producing wet sand	.169	.032	.077	.004	.064	.346
Total operating cost..	.667	.102	.217	.235	.319	1.540

Summary of costs in units of labor, power and supplies during an average month

Labor (man-hours per ton)

Stripping	0.0215
Pumping	.06
Screening	.10
Loading	.05
Track repairs	.02
Drying	.10
Supervision	.036
General	.06

Percentage of Total Cost - 59

Power and supplies

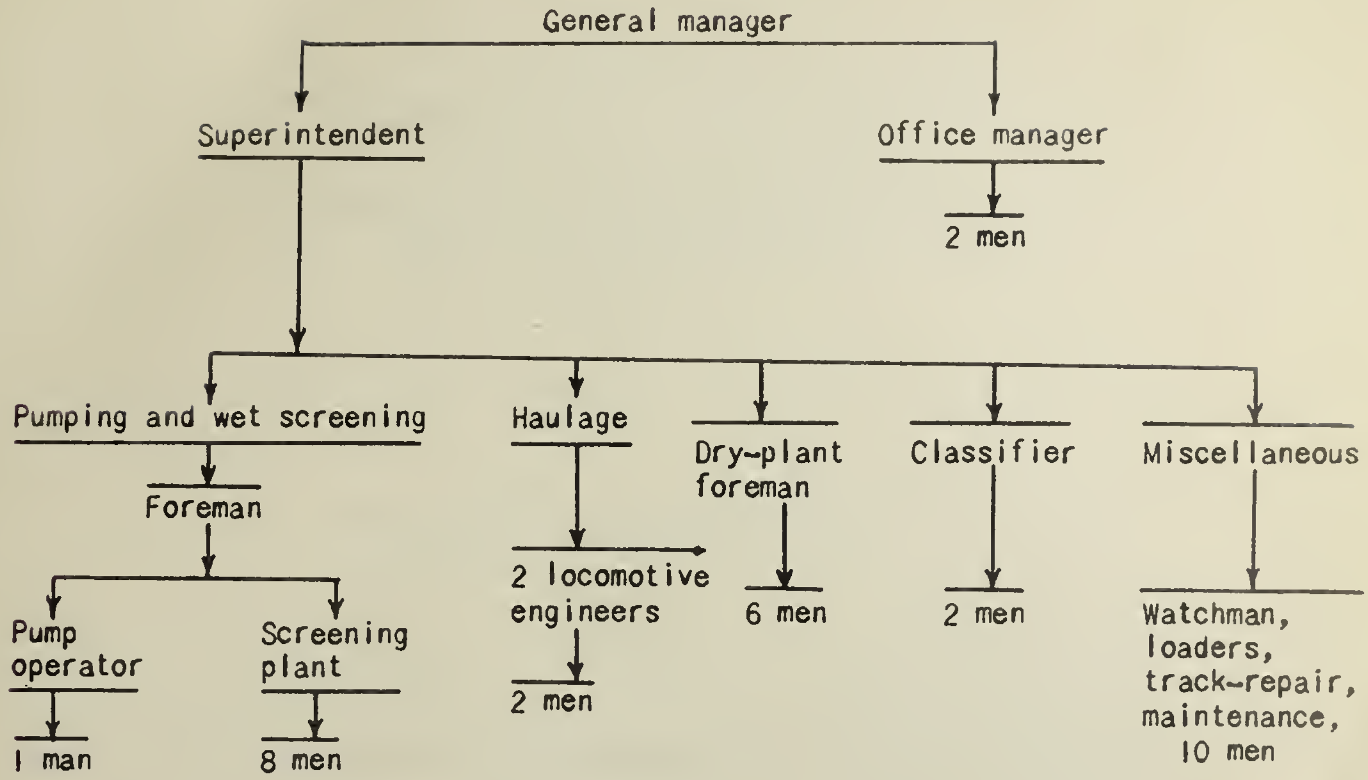
Power, kw.h. per ton

Pumping	2.04
Wet screening	.163
Drying	3.43

Cost of other supplies in percentage of total cost - 21

Power and supplies, percentage of total cost - 41.

ADMINISTRATIVE ORGANIZATION



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METHOD AND COST OF DREDGING SAND AND GRAVEL BY THE OHIO RIVER SAND CO., LOUISVILLE, KY.

BY

J. HAMILTON DUFFY

I. C. 6421
January, 1931

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METHOD AND COST OF DREDGING SAND AND GRAVEL BY THE OHIO RIVER SAND CO., LOUISVILLE, KY.[1]

By J. Hamilton Duffy[2]

INTRODUCTION

This is the first of a series of papers describing dredging methods and costs in recovering sand and gravel from the beds of rivers throughout the United States and deals directly with the methods employed and costs obtained by the Ohio River Sand Co. near Louisville, Ky.

ACKNOWLEDGMENTS

The author wishes to acknowledge the assistance of J. R. Thoenen, mining engineer of the Bureau of Mines, in the collection of data and compilation of this report.

HISTORY

The Ohio River Sand Co. operates two ladder-type dredges in recovering sand and gravel from the bed of the Ohio River in the vicinity of Louisville, Ky.

An industry which may be considered as the forerunner of the river-sand and gravel business in this vicinity was the collection of paving bowlders from the shallow parts of the river prior to 1875. These bowlders were gathered by men wading the river and scooping them up by means of forks. The bowlders were loaded into skiffs and poled or rowed to shore where they were transferred to wagons for delivery in Louisville for paving.

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6421."

2 One of the consulting engineers, U. S. Bureau of Mines, and vice president, Ohio River Sand Co.

In 1875 J. T. Duffy installed a derrick boat equipped with a clam-shell bucket. This was essentially a wooden hull on which was mounted a derrick with its steam boiler and hoist. By means of the clam-shell bucket operated by the derrick, sand and gravel were dug from the river bottom and dropped into a hopper on the derrick boat. This hopper fed directly to a short trommel with a $\frac{1}{4}$ to $\frac{1}{2}$ inch mesh screen. The sand passed through the screen and was spouted to a wooden barge moored alongside. The coarse material or gravel was discharged back into the river as a waste product.

The sand was then towed to Louisville by steam tow boats where it was unloaded by hand into wagons for delivery to the glass furnaces.

About this time the practice of sawing stone for building purposes was introduced, creating an increased market for sharp hard sand. Later on the market demands varied to include such products as mason's sand, roofing gravel, and finally sand and gravel for concrete.

As the market for river products increased, Mr. Duffy, with Paul C. Barth and James Settle, organized the Ohio River Sand Co. This company in its early years had nothing to do with the production end of the business but confined its activities to selling and distributing the material supplied by Mr. Duffy. In 1906, however, the company took over the whole operation and has continued to the present time.

In 1904 the old derrick boat and clamshell were replaced by a dredge equipped with a centrifugal pump.

This equipment was in turn replaced in 1924 with a modern ladder-type dredge to which a second unit or dredge was added in 1928. Both of these dredges are now in operation and both are described in detail in this report.

The first production of commercial gravel dates from 1904. With the derrick boat and barges unloaded by hand the company produced an average of 100 cubic yards daily which was sold at a price of 90 cents per yard, delivered anywhere in the City of Louisville.

As business increased a stiff-leg derrick was erected on the river bank, replacing the hand labor in unloading the barges. The derrick picked the sand and gravel up from the barges and dropped it into a hopper on the bank from which it passed by gravity to wagons. With this equipment the plant capacity was increased to 450 yards daily.

In 1907 the stiff-leg derrick on the river bank was replaced with a clamshell dredge. This unloaded the material into a hopper on a barge moored to the river bank. A flexible bridge and track connected this barge with an incline on shore. Wooden and steel cars now replaced the wagons and the gravel was hoisted up the incline and over a trestle to be dumped to storage piles on shore. Production was thus increased to 1,000 yards daily.

In 1923 the present land method of handling material was installed. Briefly, it consists of a Brownhoist Bridge crane and system of elevated conveyor belts delivering to truck hoppers, railway cars, or storage piles. The unloading crane is equipped with a 3-yard clamshell bucket and there are 1,500 feet of conveyors. The railway sidings have a capacity of 35 cars. There is room for storing 150,000 tons of material behind concrete retaining walls and the loading hoppers have an additional capacity of 5,000 tons.

In 1915 all wagon hauling for city delivery was replaced by auto trucks and in 1926 all wooden river equipment was replaced by steel.

The present capacity of the plant is 350 tons per hour.

GEOLOGY

The gravel deposits present the familiar characteristics of river-bar geology. Both sand and gravel have been carried down the river for centuries. Flood stages in spring and fall have brought in vast quantities of fresh material to be sorted and deposited according to the vagaries of the changing currents. Thus heavy coarse gravel is found at points of rapid current and fine sand and mud where the velocity of the current diminishes. Constant changes occurring in the direction and velocity of the currents cause the previously formed bars to be reworked and as a consequence coarse and fine gravel is found intimately mixed with large volumes of fine sand.

Formerly many of these bars were above low-water level and could be prospected and tested by dry-land methods. With the recent completion of the Federal system of dams a minimum water level is maintained which inundates practically all the river bars.

At points where the current has a comparatively low velocity the gravel is found covered with from 6 to 8 or more feet of fine sand and mud. In swifter currents coarse gravel with little fine material forms the river bed.

At many points bowlders ranging in weight from 10 to 300 pounds are found scattered over the river bed and imbedded in the gravel.

Many of the gravel bars are rendered commercially useless or at best are workable with difficulty because of water-logged debris, consisting of tree trunks, branches, and stumps carried down the river during flood periods.

During recent years an increasing amount of fine coal has been found in the deposits. It is thought this originates from sunken coal barges which have been lost in the river from time to time.

The commercial gravel deposits as found in place vary greatly both in depth and area. In a distance of 100 feet laterally a good coarse gravel deposit may change to a bar of fine sand.

In addition to the underwater deposits operated by the company it owns two islands in the river. These islands have been tested by churn drilling and the gravel beds contained in them have been shown to vary considerably. The gravel itself varies from 20 to 50 feet in vertical thickness and is overlaid with from 5 to 20 feet of sand. This in turn is covered with 5 to 20 feet of silt or river mud. The larger of these islands comprises 225 acres and is estimated to contain over 16,000,000 tons of sand and gravel.

Within the gravel beds on the islands is a stratum of heavy blue clay from 12 to 18 inches thick. This clay stratum causes trouble in that when dug up with the gravel it does not disintegrate but forms clay balls which are difficult to separate from the gravel.

In the early land grants the bottom of the Ohio River belonged to the owner of the Kentucky shore. This limit was defined as continuing under the river to the low-water mark on the Indiana shore. In most cases surveys have defined just where this low-water mark ends. The system of Federal dams in the river has in many instances permanently raised the established low-water mark on the Indiana shore by inundating considerable area. This inundation however has not changed the ownership of the flooded area. All this area between the low-water mark as established by surveys and the present shore line belongs to the Indiana owner. Therefore, Indiana land owners have been able to exploit these flooded gravel deposits by dredging since the completion of the dams.

CONDITIONS AFFECTING DREDGING OPERATIONS

At present the recovery of gravel is complicated by the condition of the river bottom as left by former dredging operations. The formerly used clamshell and pump dredges excavated shallower holes than the present type of ladder dredge. Pump suctions, as they dug, cut inverted conical openings in the bottom with tops widening as the pump went deeper. This caused caving and as the sides caved an increasing number of bowlders collected at the suction intake. This accumulation finally blocked the suction and the dredge was forced to move. This meant the gravel was recovered from a series of holes which in most cases did not extend to the bottom of the gravel. Considerable of the present dredging is over this uneven bottom from which the commercial material has been only partially removed and in which bowlders are found in troublesome accumulations. These holes have also been more or less filled by river mud and other debris depending on the length of time since they were dug.

Where undisturbed the upper 8 to 10 feet of a gravel bed will usually be found to contain 50 per cent sand and 50 per cent gravel while that below will be more apt to run 75 per cent sand and 25 per cent gravel.

The ladder type of dredge was adopted because it afforded a means of recovering a maximum of gravel with a single passage over the bar. It will also operate with less delay in bottoms that have been worked over by other types of dredges and the accumulated bowlders found therein offer little difficulty to the ladder buckets. A further reason for the choice of the ladder type over the centrifugal pump dredge is its more economical use of power. The centrifugal pump must handle from 80 to 90 per cent water with only 10 to 20 per cent solids in its delivery. This means the expenditure of a large quantity of power for moving water, which is immediately discharged to the river again. In hard-bedded gravel the gravel must often be broken up ahead of the suction by means of a cutter head. This requires further power. The ladder buckets do not require any prior cutting of the bank and deliver the gravel with a minimum of water, thus allowing the power to be expended directly for the recovery of gravel without excessive waste.

This type of dredge is necessarily of larger size and requires much larger capital investment. On the other hand the capital investment per ton of productive capacity is probably on a par with other types of dredges, while the power expenditure and labor cost per ton of material recovered are less.

No accurate data is available regarding the ratio of recovered sand and gravel to the tonnage dug. The reason for this is the variation in the material in the bars. At one place the gravel (3/8 to 2 inches) will compose 30 per cent of the material dug with roughly one per cent oversize and the balance sand. At another point the ratio may be 60 per cent gravel and 40 per cent sand and at still another point one may find 25 per cent gravel and 75 per cent sand.

PROSPECTING

Because of the erratic and constantly changing characteristics of the gravel beds prospecting in advance of actual digging is of little use. A locality prospected one year and found to contain a bed of good gravel may the year following be covered with such an accumulation of drift and debris as to make it unworkable, or the gravel may have been removed by changing river currents.

Where deposits can be reached above water level they are prospected by churn drills.

The usual practice in examining the river bottom is to sound with iron bars or pipes. By this method the operator can obtain some idea of the depth of silt or fine sand overlying the gravel or the presence of gravel itself on the river bottom.

Final prospecting, however, is done by setting a dredge over the location and actually digging. The material dug is examined for quality and ratio of sand to gravel. If not of commercial grade the dredge is moved to another locality.

METHODS OF SAMPLING

Sampling of deposits is not practiced except by visual examination of the material as brought up by the dredge buckets.

Marketable material is sampled and sent to customers for test as required.

CHOICE OF METHOD

Since all the gravel bars lie below water level and are erratic in both area and depth the only practical method of operation is some form of dredging.

The original installation of clamshell and centrifugal-pump dredges has been superseded by the present ladder dredges for reasons already discussed.

Since gravel seldom extends more than 55 feet below the water level the dredges were designed to operate at this maximum depth.

These dredges are designed and constructed to form complete, floating, washing and screening plants although no provision is made on them for storage of finished material. In addition they are provided with living quarters for their crews.

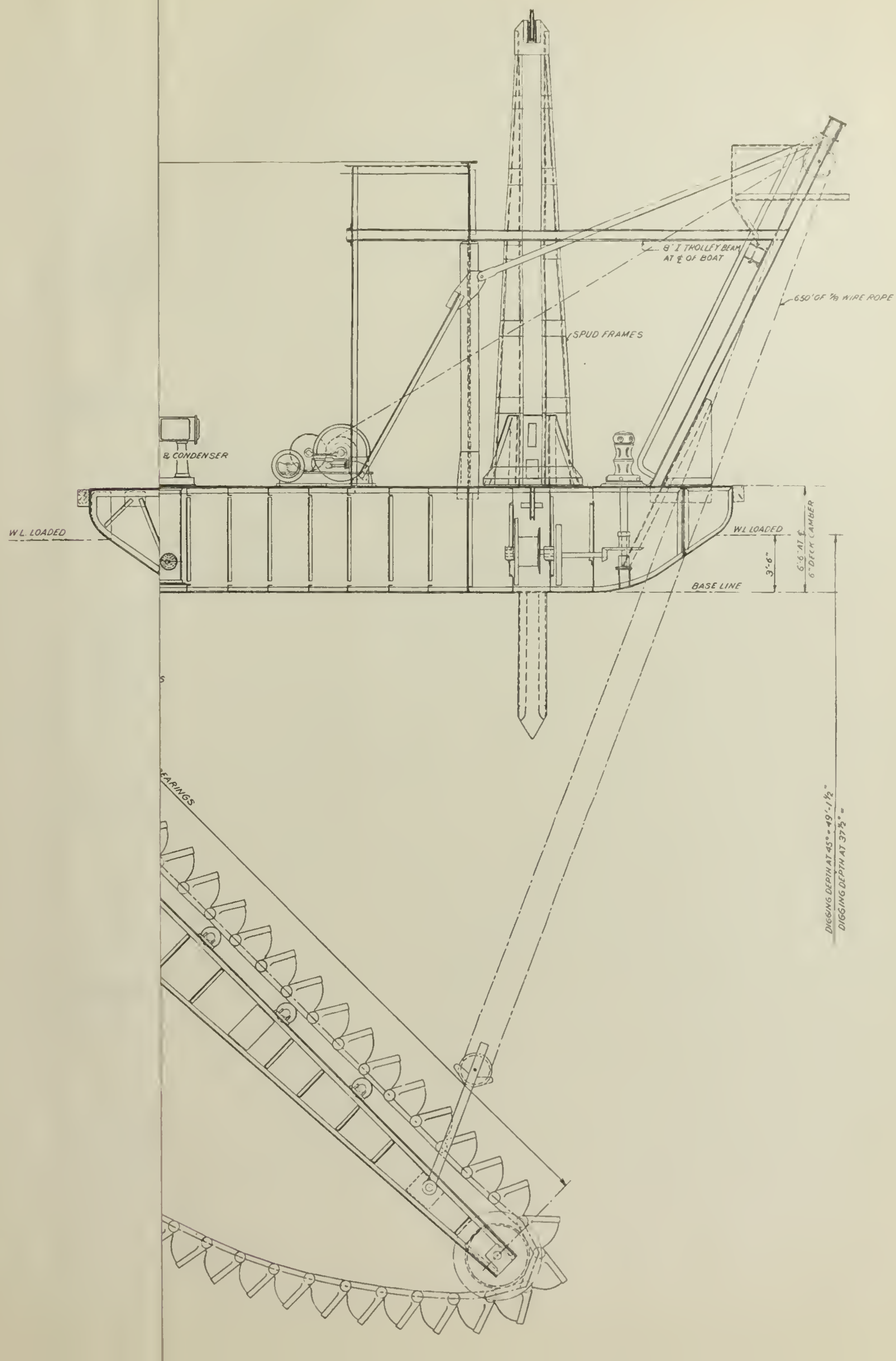
In brief, the method of operation is to dig the gravel by a rigid ladder or continuous bucket elevator. This discharges the material as dug into a hopper in which it is mixed with a sufficient quantity of wash water. It then passes to trommels wherein the sand and larger bowlders are removed from the gravel. The bowlders are discharged through chutes into the river but the sand and gravel go to separate sumps or tanks. From the sumps the gravel is raised by bucket elevator to discharge over a series of sizing screens of such aperature as the market sizes require. After passing these screens the gravel is discharged through chutes or on conveyor belts to barges moored alongside the dredge.

The sand is picked up from its sump by a second elevator and discharged to barges on the opposite side of the dredge or wasted to the river as market requirements demand.

When the barges are loaded they are towed by river steamers to the storage plant on the river bank at Louisville.

The gravel is partly washed in the digging operation. Washing continues through all the screening operations so that when discharged to the barges all silt and mud have been removed. This silt and mud pass back to the river with the excess wash water.

No further sizing is practiced at the storage plant at Louisville.


8" I TROLLEY BEAM AT ℄ OF BOAT
650' OF 7/8 WIRE ROPE
SPUD FRAMES
& CONDENSER
W.L. LOADED
W.L. LOADED
6'-6" AT ℄
6" DECK CAMBER
3'-6"
BASE LINE
DIGGING DEPTH AT 45° = 49'-1½"
DIGGING DEPTH AT 37½° =

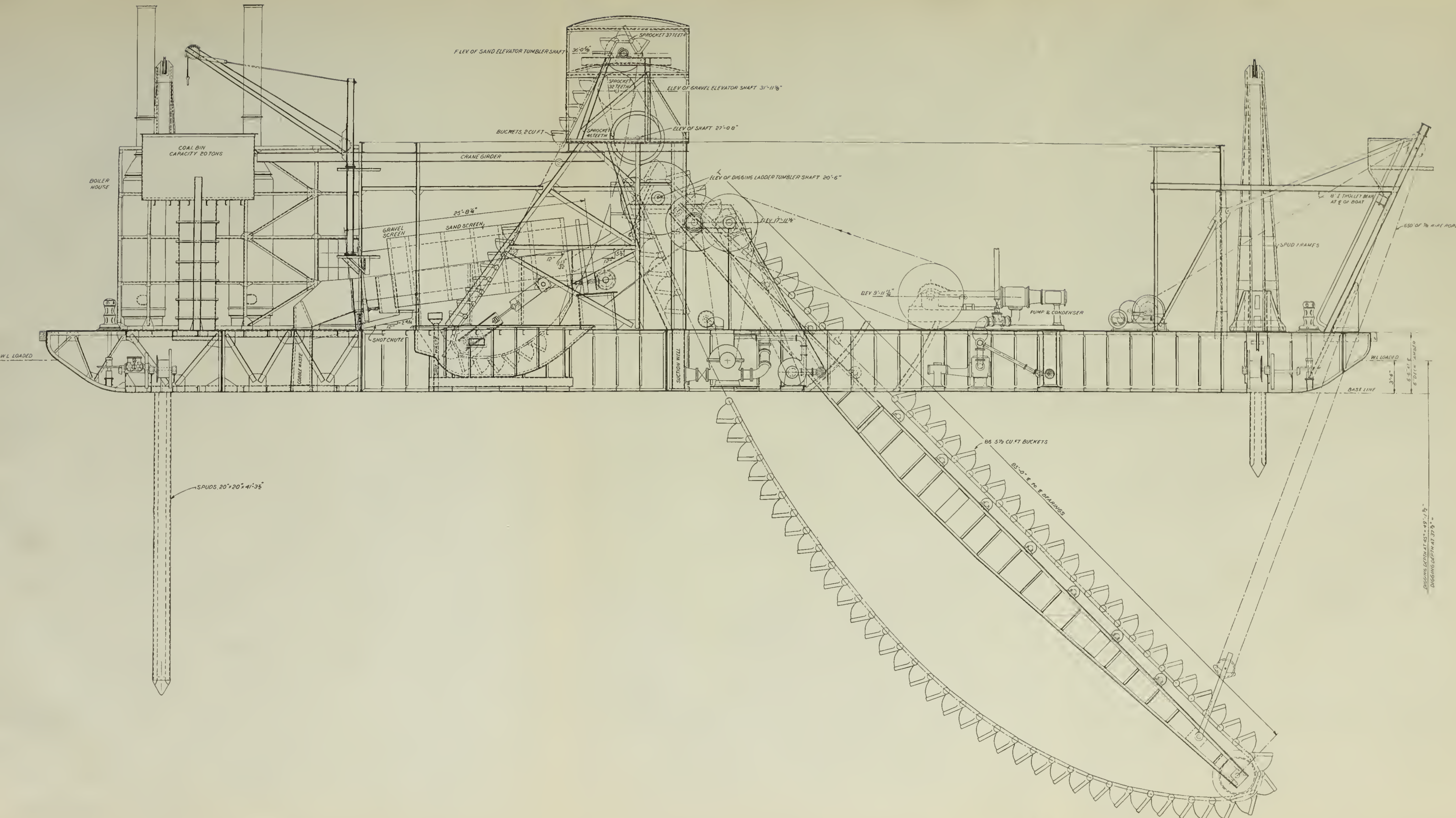
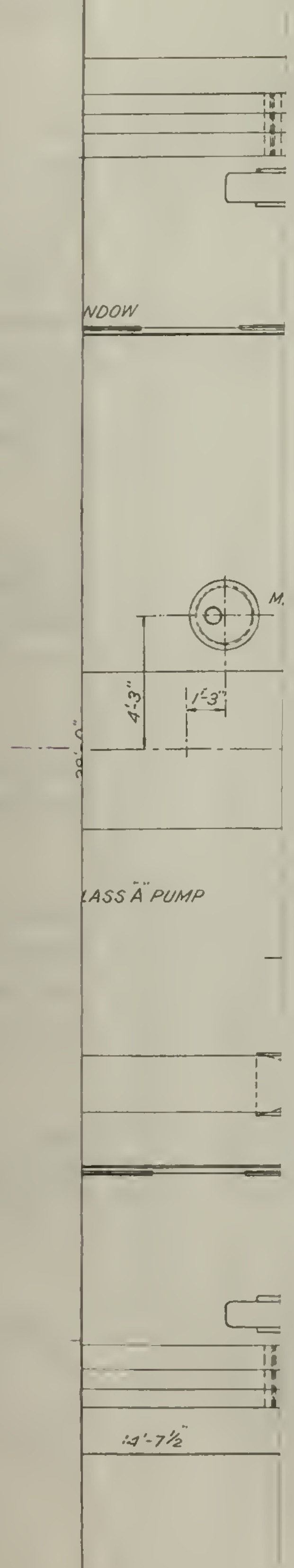


Figure I.- Elevation of No. 4 Digger


NDOW
4'-3"
1'-3"
LASS "A" PUMP
14'-7½"

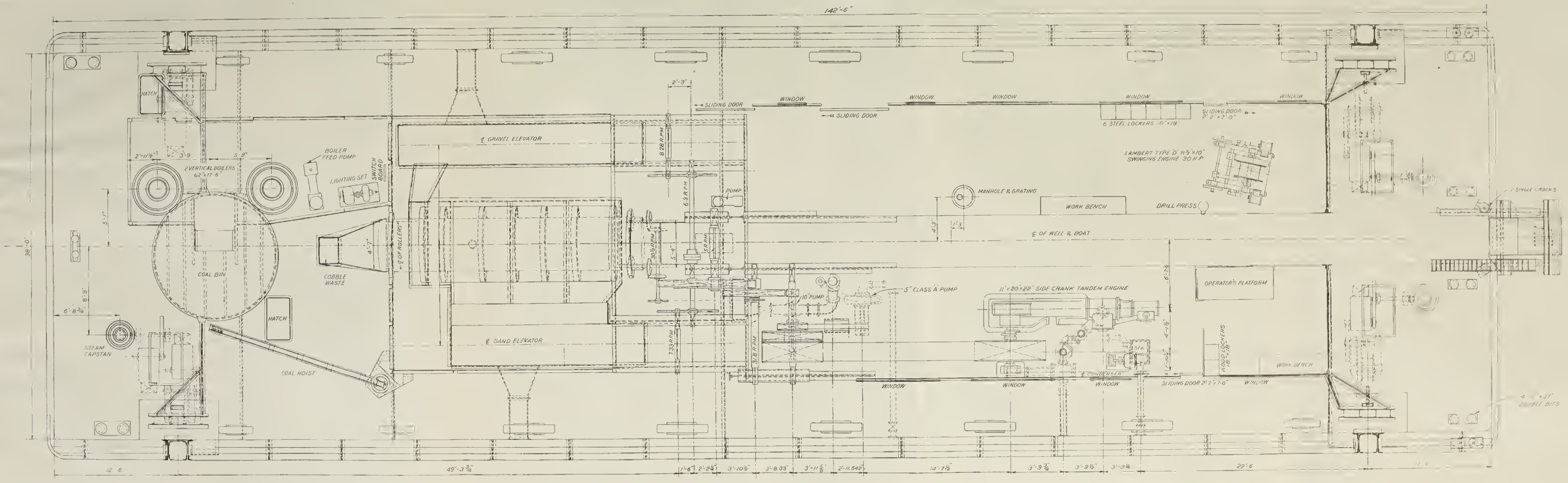


Figure 2.- Plan of No. 4 Digger

EXCAVATING

There are two dredges in operation. The older, designated as No. 4 Digger, is a steel-hulled ladder-type dredge designed and built by the Dravo Contracting Co. of Pittsburgh, Pa. An elevation and plan of this dredge are shown in Figures 1 and 2.

The digging mechanism consists of an endless bucket elevator in which the buckets themselves form the links in the chain. This elevator is mounted on a heavy rigid structural-steel frame hinged at the upper or discharge end and working through the center line of the hull. The lower or digging end is raised and lowered at the will of the dredge operator by means of multiple blocks and cable operating through a steam-driven hoist on the main deck. The operator's station is placed so he has an unobstructed view of the ascending buckets. When digging is interrupted by the buckets encountering heavy debris, such as sunken tree trunks, the operator can raise or lower the ladder by simple manipulation of the control levers.

The buckets discharge into a steel hopper supported by the steel superstructure, and the gravel is fed by gravity to a trommel having $2\frac{1}{4}$-inch round openings. Surrounding the trommel is a jacket screen with 1/8-inch openings. To prevent blinding this jacket, steel rollers are arranged to ride the outside as the screen revolves.

Oversize material which will not pass the $2\frac{1}{4}$-inch openings is discharged into the river as waste. There is not sufficient of this material found to warrant the installation of a crusher to reduce it to commercial size.

The sand and gravel passing the $2\frac{1}{4}$-inch openings but retained on the 1/8-inch screen are chuted to the boot of a continuous bucket elevator similar in construction but smaller in size than the ladder. This elevator discharges over two double-deck vibrating screens. The mesh of these screens is varied from time to time to produce the desired market sizes.

Material passing 3/8-inch round openings is designated as sand and is delivered from the vibrating screens through conveyors or chutes to the river as waste or to a barge moored alongside the digger depending on the demand for coarse sand.

Material passing the 1/8-inch jacket screen drops to a second continuous bucket elevator which discharges to a chute over the side of the digger. This discharge is also either chuted to waste in the river or to a barge as fine sand. The arrangement is such that this fine sand is discharged on one side of the digger while the coarse sand and gravel are discharged on the opposite side.

Both secondary elevators dig their material from steel tanks in which the sand or gravel is deposited from the trommel. The constant agitation of the water and sand or gravel in these tanks by the moving elevator buckets keeps the fine mud and silt in suspension, and they pass off in an overflow from the tank. The elevators thus carry comparatively clean sand or gravel to the vibrating screens or barges.

However, further rinsing is provided by additional wash water playing on the screens as the material is sized.

Ordinarily a coarse and a fine gravel are made as well as two sizes of sand.

By a system of conveyors the two sizes of gravel can be loaded into the same barge or into separate barges at the will of the operator.

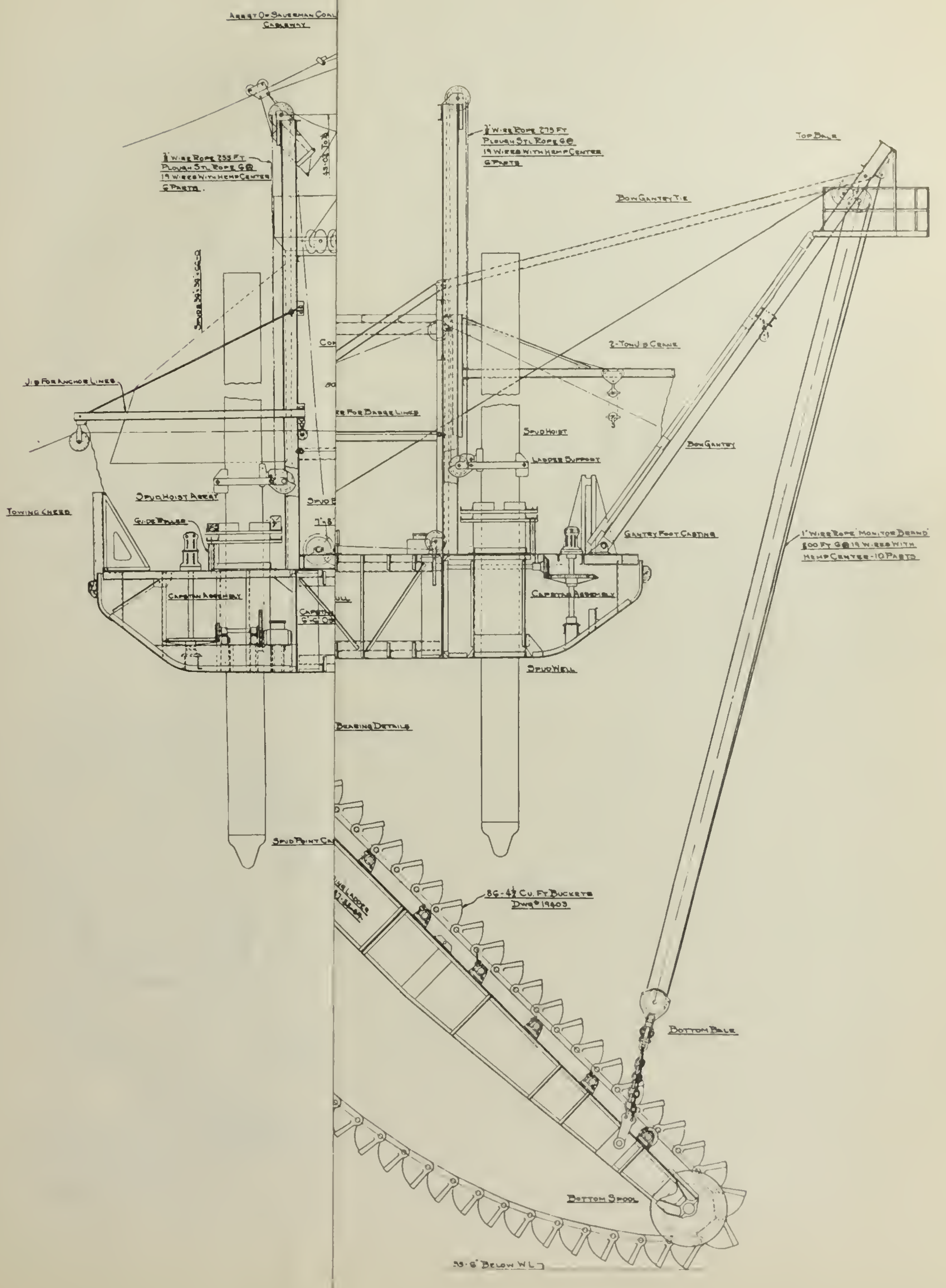
The No. 4 digger is steam driven and in average digging has a capacity of 500 tons per hour. Production at this rate requires the handling of 8,000 gallons of wash water per minute or roughly 1,000 gallons per ton of material handled.

The second dredge, known as the Kentucky, is also a steel-hulled ladder-type dredge designed and built by the Dravo Contracting Co. Since this dredge represents more modern construction it will be described in more detail, and is illustrated in Figures 3 and 4.

The hull is of 5/16-inch mild open-hearth steel plate, 155 feet long by 44 feet wide and 8 feet deep, fitted with suitable transverse and longitudinal frames, and bulkheads.

The digging ladder is built of plate and angle sections. It is 88 feet in length and provides a digging depth of 55 feet below the water line. A ten-part line running over sheaves in the bail and bow gantry provides means for raising and lowering the ladder. The ladder is fitted with 86 cast-steel buckets of 4 1/3 cubic feet capacity each. These are fitted with heat-treated steel pins and manganese-steel bushings. The buckets travel at a speed of 53 buckets per minute, corresponding to a digging capacity of 400 tons per hour.

The ladder buckets deliver their load of material to a hopper built of 3/8 and ½ inch steel plate fitted with cast-steel harp (grizzly) bars to prevent the entry of oversize material. These are spaced at 6-inch intervals and set at such an angle that bowlders will move by gravity to the lower end and fall into the waste well. Cleaning bars operated by small steel cylinders are also provided to assist the oversize material on its way to the waste well. These cleaning bars are under the control of the dredge operator.



Bow Gantry Tie
Top Bale
2-Ton Jib Crane
Spud Hoist
Ladder Support
Bow Gantry
Gantry Foot Casting
Spud Hoist Assembly
Jib for Anchor Lines
Towing Chock
Capstan Assembly
Spud Well
Bearing Details
86-4½ Cu. Ft Buckets
Bottom Bale
Bottom Spool

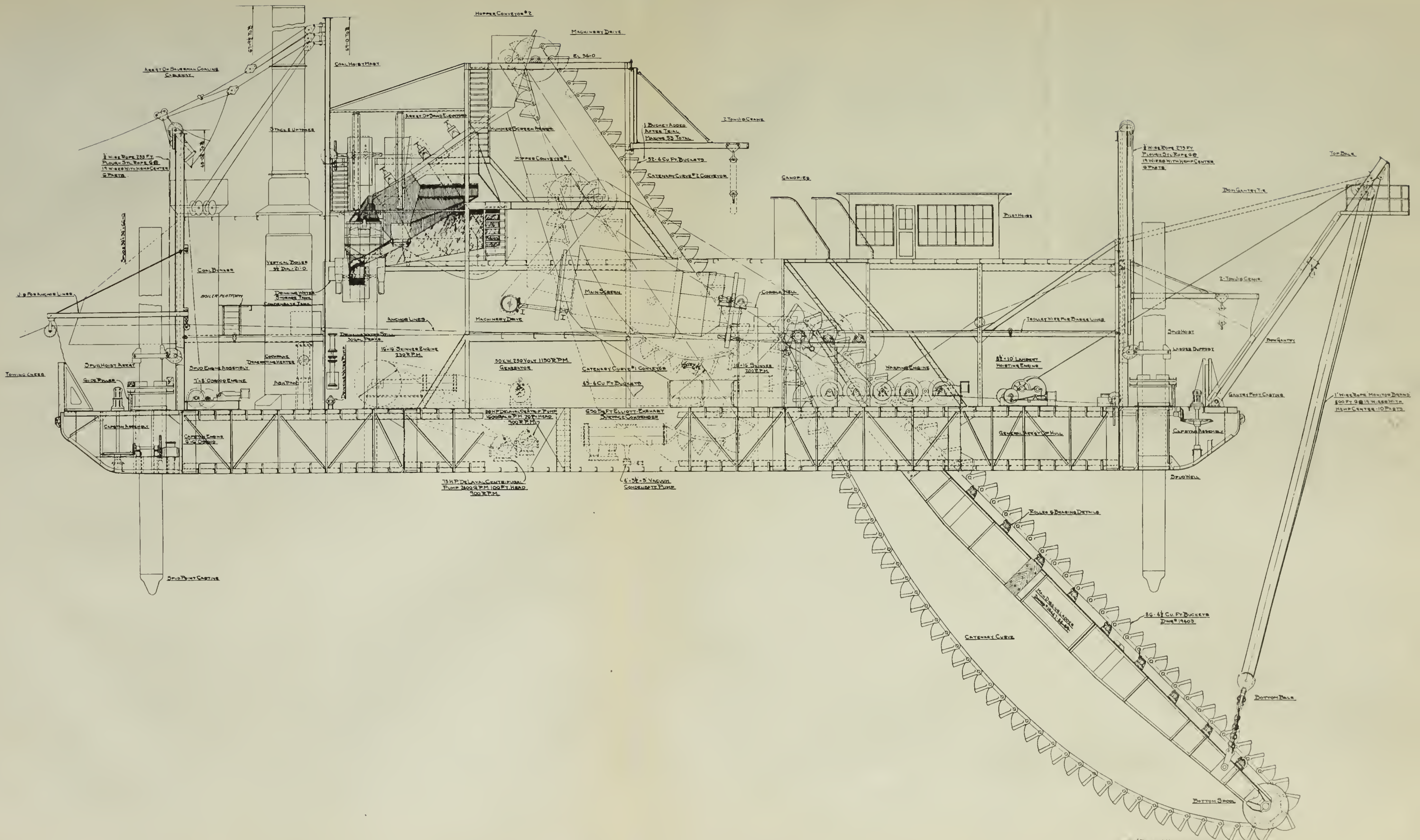
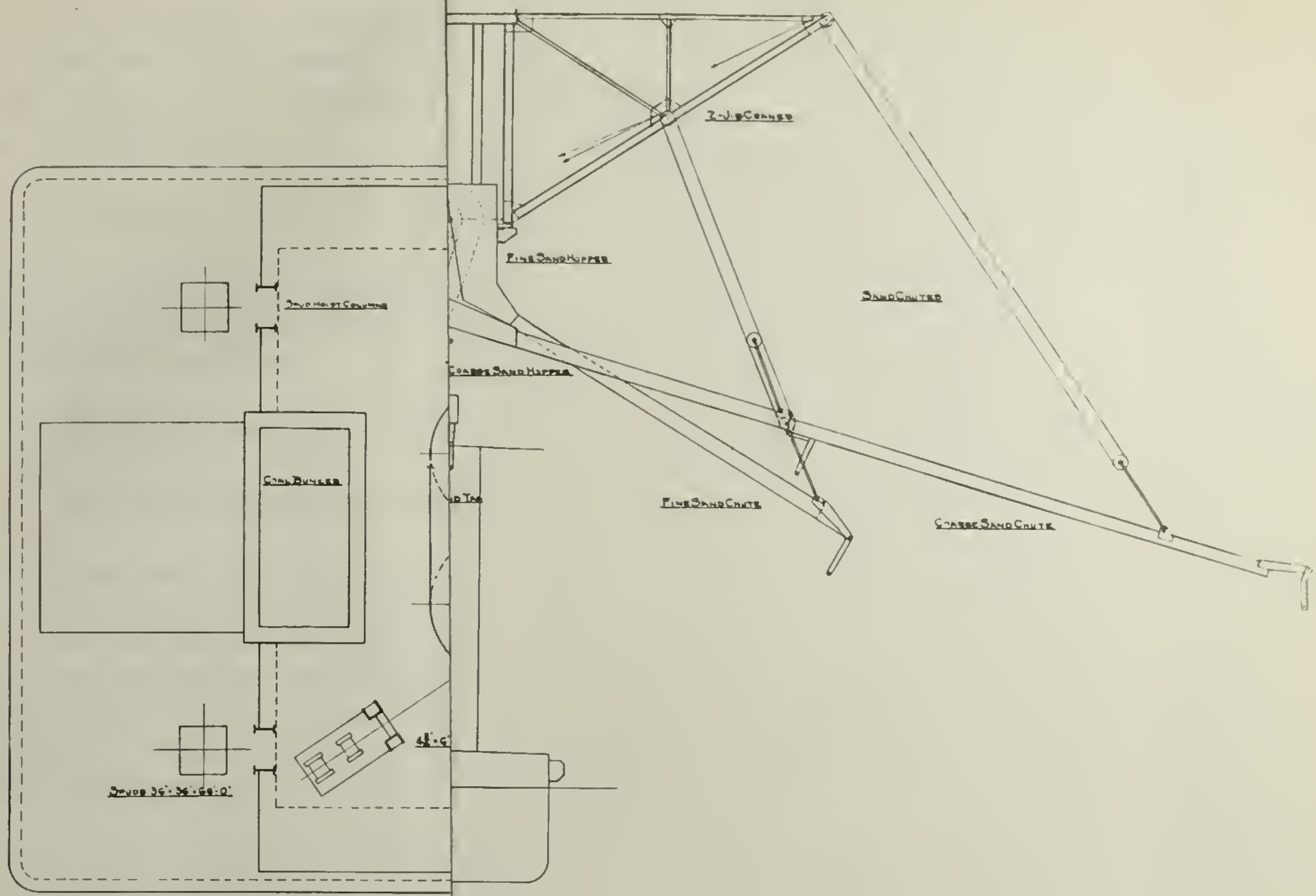
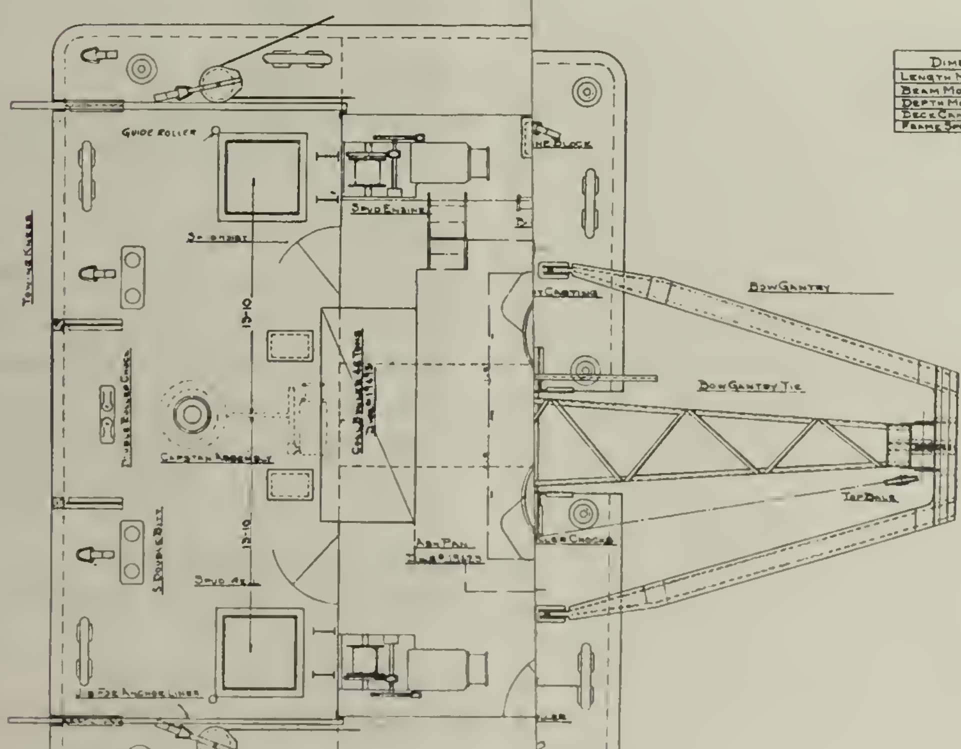


Figure 3.- Elevation of dredge Kentucky





Dimensions Of Dredge	
Length Molded	155'-0
Beam Molded	44-0
Depth Molded	[illegible]-0
Deck Camber	[illegible]
Frame Spacing	2-1

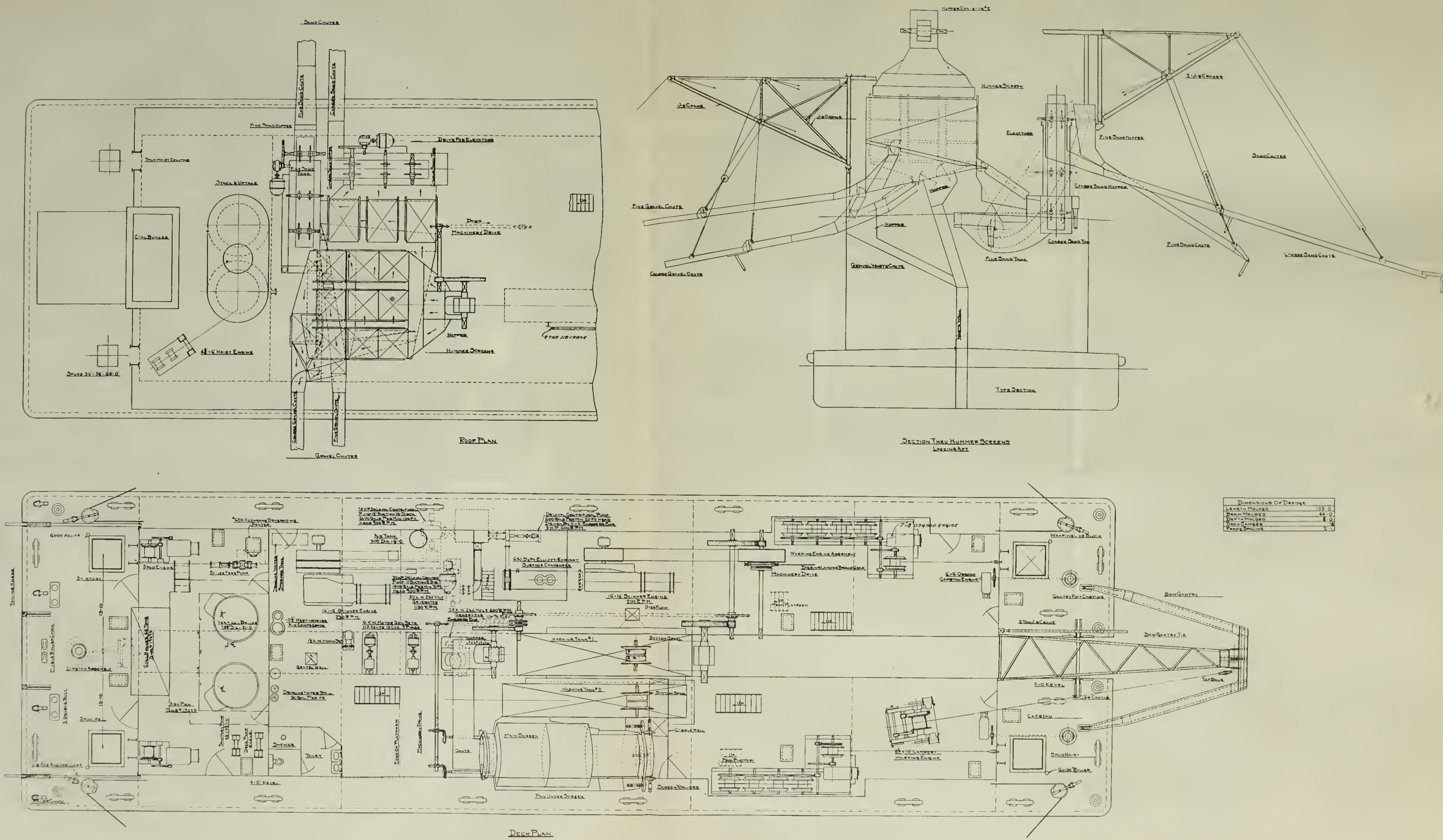


Figure 4.- Plans and section of dredge Kentucky

The material discharges by gravity to a steel washing tank from which it is elevated by a bucket elevator having 4 cubic feet buckets and discharged to a 24-foot rotary scrubber 6 feet in diameter fitted with a 13-foot jacket 9 feet in diameter.

The first 7 feet of the scrubber has $1\frac{1}{4}$-inch round perforations.

The next 6-foot section is a solid plate fitted with lifting angles for the purpose of lifting the gravel and letting it fall and thereby providing more thorough washing.

The third section, 7 feet long, has a screen with $2\frac{1}{4}$-inch perforations.

The outer screen or jacket has 5/8-inch round openings.

Wash water at high pressure is applied on all sections of the scrubber.

All material passing through the screens of the trommel passes over a flat steel pan where it is again subjected to the cleansing effect of high-pressure wash water. From this pan the sand and gravel passes to a second steel washing tank fitted with a suitable overflow for excess wash water and suspended mud and silt. This overflow discharges to the river.

Oversize material from the trommel is discharged into the river.

The washed sand and gravel in the washing tank is elevated by a bucket elevator having buckets of 4 cubic feet capacity each. The bottom spool can be raised or lowered at will. These buckets are of similar design and material as the digging ladder buckets except as to size.

This elevator delivers the material to a 3-way distributing hopper feeding 9 Tyler Hummer screens operating in parallel sets. The upper 6 screens are 3/8 by $\frac{3}{4}$ inch wire mesh and the lower three 5/8 by 5/8 inch wire mesh.

The product passing through the first 6 screens moves to 3 Tyler Hummer screens fitted with 3/16 by $\frac{1}{2}$ inch wire mesh. The oversize and undersize from this set of screens are dropped to bucket elevators and elevated to chutes discharging to barge or river as desired. That material passing over the 5/8 inch Hummer screens drops to a 4 by 8 foot Simplicity mechanically-vibrated screen, where it is drenched with final wash water before its discharge to the gravel barge.

The main conveyor drain plates are lined with 5/16-inch rubber vulcanized to 1/8-inch steel plate.

Gathering pans for sand and all sand hoppers and chutes are lined with 3/8-inch rubber vulcanized to 1/8-inch steel plate.

The two sand elevators are designed for a capacity of from 30 to 60 per cent of the digging capacity. The capacity is regulated by changing the speed of the elevator.

The main ladder engine is a 150-hp., 16 by 16 inch Uniflow condensing engine operating at 200 r.p.m. This engine transmits power through a 6-ply balata belt to the transmission shaft. The digging ladder is operated by two sets of gears from the transmission shaft. This engine drives the ladder, rotary scrubber, and two main elevators.

The rotary scrubber is operated from the main ladder drive through sprockets and chain and two pairs of bevel gears. The main conveyors are operated through sprocket and shaft and sets of spur gears.

The engine driving the high and low pressure centrifugal pumps and the generator is similar to the main ladder engine, but running at 230 r.p.m.

The two sand elevators are driven by individual electric motors through speed reducers.

Wash water is provided by a 12-inch centrifugal pump operating at 900 r.p.m. against a head of 100 feet and delivering 2,400 gallons per minute; a 10-inch centrifugal pump operating at 900 r.p.m. against a head of 40 feet and delivering 1,600 gallons per minute; and a 6-inch centrifugal pump operating at 400 r.p.m.

The first two pumps are belt driven from the auxiliary engine. The 6-inch pump is direct connected to a single-cylinder vertical Engberg steam engine with 4-inch bore and 4-inch stroke.

Steam is furnished by two vertical fire-tube boilers of 150-hp. capacity each. They are 6 by 21 feet in size and fitted with 188 2½-inch tubes 14 feet long. The boilers are designed for 170 pounds steam pressure and are provided with baffles in the upper part of the steam drum to obtain a high superheat.

The ladder hoist is a single-drum, double-reduction, reversible steam engine.

Two double-cylinder, double-gear reduction steam engines are provided for raising the stern spuds.

There are two warping engines, one on each side of the boat. They are four-drum, double-reduction, two-cylinder steam engines. These engines also drive separate drums through another set of gears for hoisting the bow spuds.

The spuds are 36 by 36 inches by 66 feet in size, built of ½-inch steel plate and 8 by 8 by ½ inch angles.

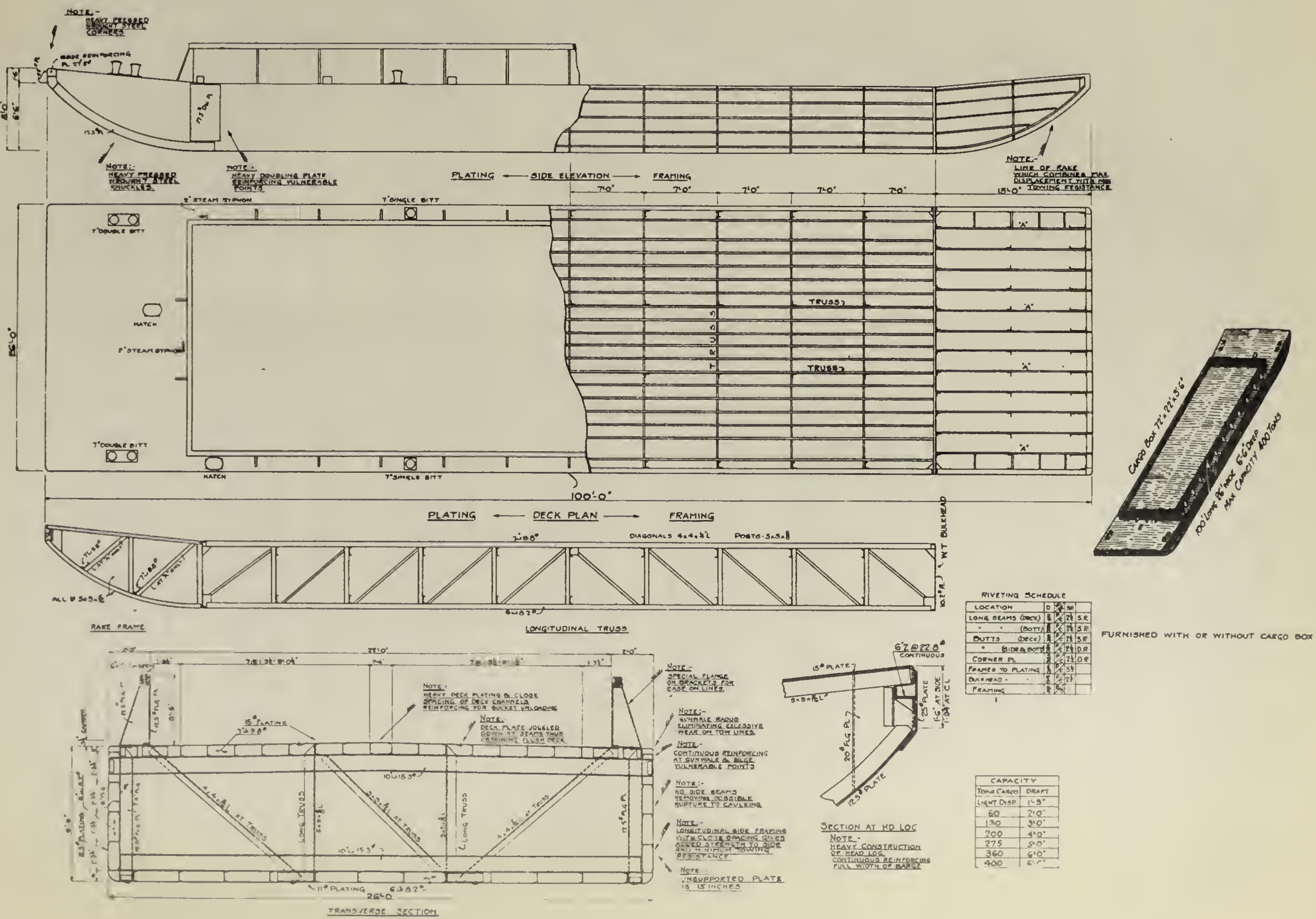


Figure 5.-Details of steel barge

The electrical equipment consists of a Westinghouse Electric 50-kw., 250-volt, 200-amp. generator operating at 1,150 r.p.m. and a $3\frac{1}{2}$-kw., 16-amp., 220-volt generator direct connected to a single-cylinder Engberg steam engine with $4\frac{1}{2}$-inch bore and 4-inch stroke operating at 505 r.p.m.

The usual complement of auxiliary machinery and equipment suitable to this type of dredge is provided, including a well-equipped but small machine shop.

Large well-equipped quarters are provided for the crew and a complete commissary and dining room is also a part of the accommodations.

TRANSPORTATION

Both No. 4 Digger and the dredge Kentucky are served by the steamer Duffy and 20 steel barges of 650 tons capacity each (fig. 5).

The Duffy is a flat-bottomed, steel-hulled, stern-wheel river tow boat, 25 by 135 feet overall with a depth from deck to bottom plates of 5 feet and a draft of $3\frac{1}{2}$ to 4 feet.

She is powered by three fire-tube marine boilers 38 inches in diameter by 26 feet long, each having six 6-inch tubes. These boilers provide steam at 240 pounds pressure to operate the driving engines and all auxiliary pumps and machinery.

The driving mechanism consists of two noncondensing Hegewald steam engines with 15-inch cylinders and $4\frac{1}{2}$-foot stroke. The pistons are direct-connected to the cranks operating the stern paddle wheel, one engine on each side.

The usual complement of boiler-feed pumps, steam winches, and capstans is present and in addition the steamer carries three light plants. One is a Pyle National, 13-ampere, 120-volt generator direct connected to a small steam turbine and is used for emergency or for such lighting as is needed during daylight.

A Delco lighting plant is provided to operate a refrigerating plant.

A Troy, single-cylinder, 8-hp., steam engine belted to a Willey Electric $8\frac{1}{4}$-kw., 120-volt, 66-amp. generator provides current for powerful search lights and general lighting for night operation.

The steamer is provided with Gardner steam-driven steering gear.

The Duffy can pull 10 loaded barges down stream at a speed of about 3 miles per hour. Running light she has a speed of about 8 miles per hour.

A full double-shift crew consists of 13 men but only 6 men are required for daylight running.

Quarters are provided on board for the crew and in addition a complete kitchen and boarding equipment.

STORAGE PLANT

Sand and gravel brought from the dredges in steel barges is moored to the river bank at Louisville, while the tow boat returns the empty barges to the dredges.

Loaded barges are manipulated under the unloading equipment by means of a double-drum Thomas hoist engine installed on a steel-hulled landing barge or pump boat. This hoist is operated by steam at 100 pounds pressure furnished by a 150-hp., marine-type, water-tube boiler also located on the pump boat. This boat is kept moored at the landing and serves as a landing dock or floating wharf over which supplies are moved to the tow boats. As the name implies, it also supplies power for pumping water from leaking or flooded barges. For this purpose steam from the boiler is used in steam-ejector pumps.

By means of the double-drum hoist loaded barges can be moved in either direction, spotted in position for unloading, and moved as unloading proceeds. The empties are removed and replaced by loaded barges by the same power.

Gravel is unloaded by a Brown-Hoist bridge crane equipped with a $3\frac{1}{2}$-cubic yard clamshell bucket. This crane is of steel construction, cantilever type, with an overall reach of 200 feet. The crane is stationary and all barges must be spotted under it.

Paralleling the river and underneath the crane is a railroad spur track on which cars are spotted for direct loading from the barge when necessary. Ordinary procedure, however, involves discharging the crane bucket to a steel hopper of 25 cubic yards capacity located between the spur and the river bank.

The hopper delivers the material to a 36-inch rubber conveyor belt through a 48-inch steel pan feeder operated through an eccentric. This belt travels at 240 feet per minute and discharges either direct into railway equipment on the spur after passing over a weightometer, or over a shaking screen with $\frac{1}{4}$-inch slotted openings to a second 36-inch rubber conveyor belt.

The first belt is about 75 feet long and operates on an incline of 18 per cent. When discharging over the screen to the second conveyor the gravel is subjected to a final rinsing by a stream of water playing on the gravel as it passes over the screen. The screen is actuated by an eccentric driven from the head-pulley shaft of the first conveyor. Wash water and sand are returned to the river. The initial conveyor is driven by a 10-hp. electric motor operating on 440-volt, alternating current.

The second or main conveyor is 36 inches wide and 600 feet long and runs up an 18-per cent grade for 75 feet and level for the balance of its travel except as it rises to pass over the tripper. This belt is driven by a 35-hp. motor.

This main conveyor delivers through a double-discharge tripper to either of two shuttle conveyors set at right angles to the main conveyor.

The shuttle conveyors are driven by 12-hp. motors. The belts are 36 inches wide and 125 feet long and are reversible in direction. Running in one direction No. 1 shuttle delivers through a movable tripper to a series of 7 concrete bins each of which holds 500 tons. These are used as gravel bins and separate such gravel sizes as have been made on the dredge. No attempt is made to resize or grade either sand or gravel except on the dredge.

Delivery from these bins is to auto trucks through hand-operated flat horizontal gates.

Operating in reverse direction this shuttle conveyor delivers gravel to a ground storage pile.

The No. 2 shuttle conveyor is also reversible in direction, delivering to ground storage in one direction and to 6 concrete bins of 350 tons capacity each in the other direction. This shuttle is used to deliver sand to the bins or gravel or sand to storage, as may be required. The bins store the fine, medium, and coarse sand as delivered by the dredge and discharge to auto trucks.

Either of these shuttles can discharge direct to railway cars on a spur running below them and paralleling the main conveyor.

The No. 2 shuttle conveyor also delivers material beyond the sand bins to a fifth conveyor 24 inches wide which in turn delivers to either of two 24-inch by 260-foot storage conveyors which discharge to ground storage.

In reverse direction this shuttle delivers to an eighth conveyor 24 inches by 150 feet, which also discharges to ground storage.

Either sand or gravel may be handled by these conveyors as exigencies may require.

Centrally located near the main conveyor is the control room from which all belts may be operated or controlled. All belts may be instantly stopped by push-button control from the control room or from points located at 30-foot intervals throughout the system. The first belt carrying material from the hopper under the bridge crane may be operated either from the crane or from the control room.

All belts are driven by separate motors operating on 440-volt, alternating current.

The two long storage belts are run by 25-hp. motors and the short storage belt is operated by a 15-hp. motor.

All 36-inch belts operate at a speed of 240 feet per minute and the 24-inch belts at 320 feet per minute.

The system has a capacity of 350 tons per hour and is lighted so it can be operated day or night.

The yard has a capacity of 150,000 tons of sand and gravel in ground storage and in addition the railway spurs will hold 35 60-ton cars (see fig. 6).

Cars spotted on the river spur are moved by a double-drum Thomas elevator hoist and $1\frac{1}{2}$-inch cable controlled by the crane operator. This hoist is powered by a 75-hp. motor and can handle 15 loaded cars at one time. All chutes discharging material from trippers are lined with rubber. These rubber linings have been found to outlast an equal thickness of steel plate many times.

A gasoline-driven caterpillar crane loads railway equipment from storage piles. This crane is equipped with a $\frac{1}{2}$-yard clam-shell bucket.

For wagon or truck loading from storage the yard is equipped with 4 Barber-Green gasoline-driven portable loaders mounted on caterpillar traction. These loaders are capable of delivering from storage 1 ton per minute each.

There is also provided a Green slip scoop or drag bucket of $1\frac{1}{4}$ cubic yards capacity for loading railway cars from storage. This is equipped with a Thomas elevator hoist powered by a 75-hp. motor.

The bridge crane is also used in transferring heavy machinery from shore to river equipment and has a capacity of 10 tons. The crane is powered by three separate motors, one for hoisting, one for closing and one for travel. All are of 150 hp. and interchangeable.

For city delivery the company maintains a fleet of 10 Mack and White 5-ton trucks, all of which are garaged in one corner of the yard which is 600 by 800 ft. in size and entirely floored with concrete.

In one end of the garage is a completely equipped machine shop wherein all equipment repairs are made as needed.

In unloading barges three men are employed to clean up after the crane bucket.

All storage piles are confined behind 9-foot concrete retaining walls and the conveyor supports are reinforced concrete with supporting spans of structural steel.

As will be noted in the above description, no provision is made at the land plant for rescreening or grading either sand or gravel. Material is merely transferred from barges to storage bins or railway equipment.

All sizing and grading is done on the dredges and ordinarily two sizes of gravel and three of sand are made. The sand, however, is graded as to color also. Some of the deposits produce a gray sand and others a red sand. These are handled and stored separately for marketing purposes.

Personnel and wage rates

Storage-yard crew:

1 crane man, caterpillar	$6.50
1 hopper tender	4.00
1 car cleaner	5.00
1 wagon loader	6.00
3 barge tenders	4.00
1 operator, bridge crane	7.50
1 belt operator	5.00
1 electrician	7.50
1 car tender	4.00

Steamer Duffy:

1 pilot	10.00 and board
1 engineer	7.00 and board
1 mate	4.75 and board
2 firemen	4.75 and board
1 cook	3.00 and board

Dredge Kentucky:

1 operator	8.00 and board
1 engineer	7.75 and board
1 fireman	4.50 and board
2 deck hands	4.00 and board
1 mate	4.50 and board
1 night watch	4.00 and board
1 cook	3.00 and board

No. 4 Digger:

1 operator	$8.00 and board
1 engineer	7.75 and board
1 oiler	4.00 and board
1 fireman	4.50 and board
2 deck hands	4.00 and board
1 mate	4.50 and board
1 night watch	4.00 and board
1 cook	3.00 and board

COSTS

Period Covered - January 1 to December 31, 1927.

Total material loaded:

Digger No. 4	743,712 tons	(Ladder dredge)
Digger No. 3	73,554 tons	(Centrifugal-pump dredge)
Total dug	817,266 tons	
Bridge crane	634,596 tons	(To storage)

Operating Costs Per Dry Ton Produced

	Labor		Power		Supplies		Depreciation		Total	
	Total	Per ton	Total	Per ton	Total	Per ton	Total	Per ton	Total	Per ton
Digging #3 Digger	$ 7,061.85	$ 0.096	$ 1,692.72	$ 0.023	$ 4,168.51	$ 0.057	$ 2,974.75	$ 0.040	$ 15,897.83	$ 0.216
#4 Digger	22,424.75	.030	10,759.28	.015	36,209.86	.049	22,036.57	.029	91,430.46	.123
Towing Str. Duffy	11,480.05	.014	7,736.71	.009	3,377.80	.004	5,753.03	.007	28,347.59	.034
Storage	18,841.30	.030	4,050.00	.006	30,727.35	.048	11,234.58	.018	64,853.23	.102

Costs are shown for the year 1927 rather than more recent figures because during that year the dredging equipment had a minimum of lost time and operating costs are therefore more truly representative of the capability of the different types of equipment.

No. 3 Digger was a centrifugal-pump dredge operating a 10-inch pump. This unit has since been abandoned and the Kentucky substituted.

Summary of Costs in Units of Labor

Period covered - January 1 to December 31, 1927.

	Digging	Towing	Storage
Labor (man-hours per ton)			
No. 3 Digger	0.12		
No. 4 Digger	.036		
Total digging	.044		
Steamer Duffy		0.022	
Storage yard			0.061
Total man-hours per ton dug	.114		
Power			
Kw.h. per ton			0.255

Labor, per cent of total cost	30.0
Supplies and power, per cent of total cost	49.3
Depreciation, per cent of total cost	20.7

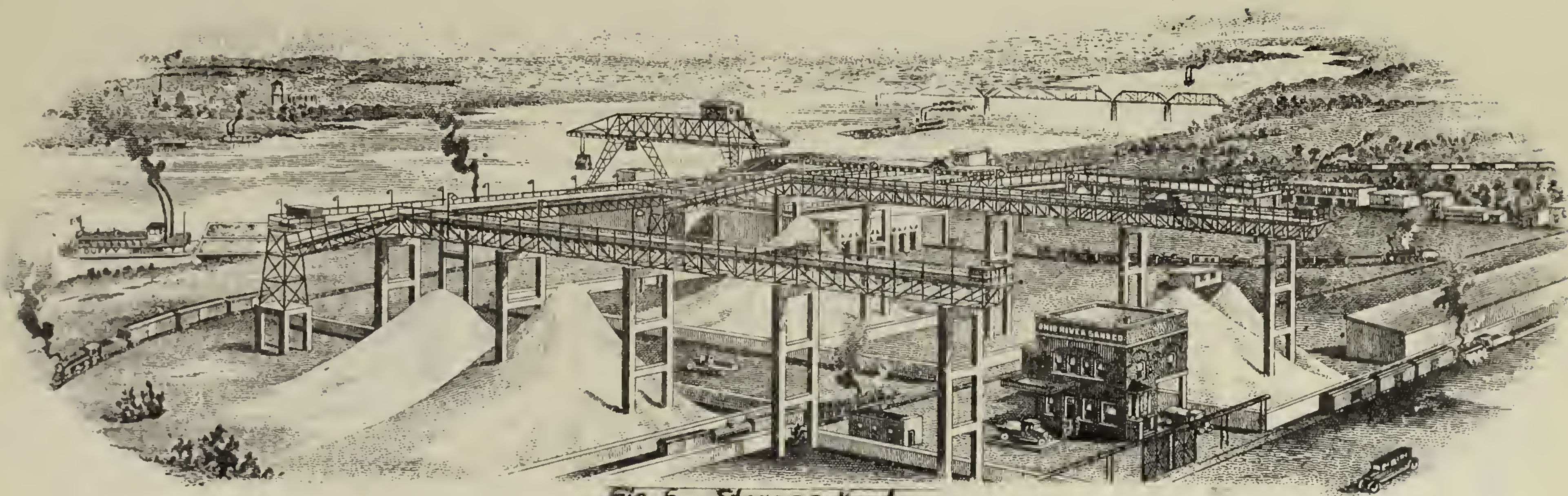


Fig. 6. Storage Yard.

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I. C. 6422

December, 1930

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES

SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

GEOPHYSICAL ABSTRACTS

NO. XX

BY

FREDERICK W. LEE

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

GEOPHYSICAL ABSTRACTS[1]

No. 20

Compiled by Frederick W. Lee[2]

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6422."

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1. GRAVITATIONAL METHODS

METHODS OF REDUCING GRAVITY OBSERVATIONS

By Walter D. Lambert

Gerlands Beiträge zur Geophysik, vol. 26, No. 2, 1930, pp. 185-188

The methods for reducing gravity observations will vary according to the purpose for which the results are to be used. For a land station Prey's method, recommended by Hopfner, would give approximately the actual value of gravity at a point within the crust directly under the gravity station. A system of values of gravity thus derived would, however, be useless either for studying the inhomogeneities of the earth's crust or for determining the figure of the earth. For these two purposes values of gravity derived from the potential for external space are needed.--Author's abstract.

DIE HYPOTHESENFREIE REDUKTION UND NUMERISCHE VERARBEITUNG DER BEOBACHTETEN SCHWERKRAFTWERTE

(THE AHYPOTHETICAL REDUCTION AND NUMERICAL WORKING UP OF THE OBSERVED GRAVITY VALUES)

By F. Hopfner

Gerlands Beiträge zur Geophysik, vol. 25, No. 4, 1930, pp. 339-347

From the results of gravity measurements and in connection with the formulas of Clairaut and Stokes, the face of an earth plane at sea-level may be determined synthetically without the aid of the hypothesis of distribution of masses inside the earth crust. Disputes about the isostatical distribution of masses inside the earth crust might be decided by computing with this method because the amount of the undulation becomes known.--Author's abstract.

UEBER EINE ABSTIMMVORRICHTUNG DER HALBSEKUNDENPENDEL STÜCKRATHSCHER FORM

(ARRANGEMENT FOR ADJUSTING HALF-SECOND PENDULUMS OF STÜCKRATH'S TYPE)

By M. Rössinger

Zeitschrift fuer Instrumentenkunde, vol. 50, No. 9, 1930, pp. 552-554.

In order that the "process of phase-opposition oscillations" (Gegenschwingungsverfahren) may be used successfully during practical pendulum measurements, it is desirable that the periods of oscillations of two pendulums swinging in opposite directions coincide as much as possible. Therefore the author thought it necessary to provide the pendulums with an arrangement by which, in addition to the adjustments made by the manufacturer, the observer can make accurate adjustment also.

Such an arrangement introduced into the four-pendulum apparatus constructed by the Askania-Werke in Bamberg is described in this article.--W. Ayvazoglou.

UEBER DEN EINFLUSS DES ERDMAGNETISCHEN FELDES AUF DIE SCHWINGUNGSZEITEN VON NICKELSTAHLPENDELN

(THE INFLUENCE OF THE EARTHMAGNETIC FIELD ON THE TIME OF OSCILLATIONS OF NICKEL STEEL PENDULUMS)

By M. Rössinger

Zeitschrift fuer Instrumentenkunde, vol. 50, No. 9, 1930, pp. 551-552

In this article the author examines the influence of the earthmagnetic field on the time of oscillations of nickel-steel pendulums and gives the results of this examination.--W. Ayvazoglou.

2. MAGNETIC METHODS

UEBER DIE MAGNETISCHE EIGENSCHAFT VON GESTEINEN

(ON THE MAGNETIC PROPERTY OF ROCKS)

By J. G. Koenigsberger

Terrestrial Magnetism and Atmospheric Electricity, vol. 35, No. 3, 1930, pp. 145-148.

The topographical magnetic effect in the Alps and in the Black Forest are given approximately by the magnetic induction calculated from the observed susceptibility; the same is true according to Carlheim-Gyllensköld for anomalies caused by the magnetic ore of Lapland. But Folgheraiter, Brunhes, David, and Chevallier could explain their observations only by remanent magnetization of the rocks studied. The author has therefore studied this question and found that all rocks have remanent magnetization, induced during cooling between 450 and 600° C., which is for intrusive igneous rocks about 0.2 to 0.8 of the induction (J_k) calculated from the observed susceptibility and the total intensity at this time. Many intrusive rocks, especially lavas, have a higher ratio --from 1 to 10. This high ratio must be explained by a remanent magnetization near saturation and a large coercive force; the available inductive susceptibility at 20° C. is now smaller than was the corresponding susceptibility at the higher temperature above 400°. The variability of this ratio suggests a variability of some physical conditions, for example, movements, during the time of cooling at the temperature at which magnetization becomes remanent. The average influence of the remanent magnetization of intrusive and of many extrusive rocks at some distance is often very small, while the direction of the magnetization is changing rapidly, as was proved by measurements on different rocks. The absolute value of susceptibility (K) and of the remanent magnetization (J) is often not constant in a cube of 4 to 7 centimeter edge.

This inhomogeneity u_1, u_2, u_3, and the anistropy a_1, a_2, a_3, were also measured for the three directions of a cube of the rock.--Author's abstract.

EARTH MOVEMENTS AND TERRESTRIAL MAGNETIC VARIATIONS

By Ross Gunn

Terrestrial Magnetism and Atmospheric Electricity,
vol. 35, No. 3, 1930, pp. 151-156

It is shown that a general contraction of the earth or a movement of a small part of it may give rise to appreciable disturbing magnetic fields. The movement of a conducting region inside the earth across the earth's magnetic field sets up electromotive forces which may produce large current systems. These current systems can give rise to the observed magnetic variations if the conductivity of the earth's core is sufficiently large.

Earlier determinations of the resistivity of the earth's core are found to be to several orders of magnitude too high, due to the neglect of skin effects. It seems probable that the electrical time-constant of the earth's core is not less than 1,000 years, and therefore magnetic diurnal variation data can not be used to calculate its conductivity.--Author's abstract.

EARLY DECLINATION OBSERVATIONS, KAMCHATKA TO BERING STRAIT

By W. N. McFarland

Terrestrial Magnetism and Atmospheric Electricity,
vol. 35, No. 3, 1930, pp. 161-164

This article contains a list of measurements of the magnetic declination at 23 points along the Siberian coast from Kamchatka to Bering Strait, as observed by the officers of Bering's expedition of 1728 and 1729, together with a declination chart of the region based on the observations.--Author's abstract.

LATEST ANNUAL VALUES OF THE MAGNETIC ELEMENTS AT OBSERVATORIES

By J. A. Fleming

Terrestrial Magnetism and Atmospheric Electricity,
vol. 35, No. 3, 1930, pp. 165-177

Annual values of the magnetic elements observed at a very large number of observatories (91) scattered all over the world are compiled by the author.

In the remarks accompanying the list the author notices that the uneven distribution of magnetic observatories is strikingly evident and in particular the lack in the southern hemisphere, especially in Africa and the oceanic areas. Locations in which fully equipped observatories, which would contribute greatly to the needs of research in terrestrial magnetism, should be established are mentioned.--W. Ayvazoglou.

DAS NEUE OBSERVATORIUM IN NIEMEGK

(THE NEW OBSERVATORY IN NEIMEGK)

By A. N. Nippoldt

Terrestrial Magnetism and Atmospheric Electricity,
vol. 35, No. 3, 1930, pp. 180-181

Tells briefly of the installation of a new observatory in Niemegk with its directors remaining in Potsdam. The old buildings in Potsdam are transformed into geomagnetic laboratories. The new observatory received the name "Adolf Schmidt Observatory for Earth Magnetism" in commemoration of Prof. Adolf Schmidt, whose seventieth birthday was celebrated on the day of the opening of the observatory.--W. Ayvazoglou.

SUR LES PROPRIÉTÉS MAGNÉTIQUES DES ROCHES

(ON THE MAGNETIC PROPERTIES OF ROCKS)

By G. Grenet

Annals Physique, Paris, vol. 13, March, 1930, pp. 263-348

An abstract of this article is published by H. Harradon in the Terrestrial Magnetism and Atmospheric Electricity, vol. 35, No. 3, 1930, p. 183. The abstract reads as follows:

> The first part of the paper is devoted to a historical sketch in which the author passes in review the work of the principal investigators who have occupied themselves with the study of the magnetization of rocks and summarizes their individual contributions to our knowledge of the subject. He then comments critically on the various methods employed for determining the magnetic susceptibility of rock samples and gives a detailed description of the two instruments selected for his own measurements, namely, a Curie-Cheneveau balance with certain modifications and a Hughes induction balance especially adapted to the rapid determination of the coefficient of magnetization of rock samples of moderate size. The use of this latter instrument is justified on the ground that the eddy currents induced in the samples do not vitiate the measurements to an appreciable extent.
>
> A detailed exposition and discussion of the results of the investigations are given, from which the author draws the following conclusions: (1) Although the classification of rocks used in petrography does not conform to a classification of the magnetic properties, rocks rich in ferromagnesian constituents are in general more magnetic than rocks poor in these elements; (2) an examination of thin sections by a polarizing microscope shows that the values of the susceptibility observed are explained by the magnetic content of the respective samples; (3) in the present state of our knowledge, it may be assumed that magnetite is the principal magnetic constituent of

rocks although the experiments of various investigators seem to prove that certain rocks contain other magnetic substances which are revealed by their Curie points; (4) the experiments of A. Michel-Levy and the author show the relations observed between the increase of susceptibility of certain rocks after heating and the changes brought about in their constituent minerals; (5) rocks of fluidal texture have a slight magnetic anisotropy.

The investigations described were made at the Institut de Physique du Globe of the University of Paris.

A bibliography containing 64 titles concludes the paper.--H. D. Harradon.

BESTIMMUNG MAGNETISCHER SUSZEPTIBILITÄTEN VON GESTEINEN UND MINERALIEN IN SCHWACHEN MAGNETISCHEN FELDERN

(DETERMINATION OF MAGNETIC SUSCEPTIBILITIES OF ROCKS AND MINERALS IN WEAK MAGNETIC FIELDS)

By J. Koenigsberger

Centralblatt fuer Mineralogie, Geologie und Paläontologie, 1929, Abt. B, No. 4, pp. 97-107

In this article the author discusses the application of a method by which the determination of the susceptibility of rocks and minerals in weak magnetic fields is possible and which is especially useful if many samples, not all of which can be brought to the laboratory, are to be investigated. The method is explained on the basis of examples and illustrated by figures.--W. Ayvazoglou.

UEBER DAS MAGNETISCHE VERHALTEN VERSCHIEDENER HARZGESTEINE

(ON THE MAGNETIC BEHAVIOR OF VARIOUS HARZ-ROCKS)

By H. Reich and W. Wolff

Centralblatt fuer Mineralogie, Geologie und Palaontologie, 1929, Abt. B, No. 5, pp. 153-161

In this article the authors give the results of their investigations made on the magnetic effect of various Harz-rocks. These investigations may serve as a basis for future work. The most important result of the investigation consisted of the fact that a relatively strong magnetic effect of the red hematite deposit in the central Harz was proved. The magnetic profile given in the article shows a surprising conformity with the known geological conditions of the region.

The results of investigations made by the authors upon various Harz-rocks are summed up as follows:

1. The Paleozoic sediments of the Harz are magnetically very weak. The susceptibility of Wissenbacher slate is slightly higher and that of Culm sediments is lower than the susceptibility of the other layers.

2. The behavior of the deep rocks of the Brocken and Ramberg is varied. The granites of the Ramberg massif are magnetically very weak and do not differ greatly from the sediments surrounding them. On the other hand, the diorites and perhaps one part of the border granites of the eastern zone of the Brocken massif are magnetically very strong.

3. Eruptive rocks and dike rocks. The effects of diabases and keratophyres are, of course, not as strong as those of the deep-rocks mentioned above, but sometimes the effects are very marked, especially in case of coarse-grained diabases. The meso-volcanic dike rocks are magnetically very weak.

4. The red hematite deposits have shown very high magnetic disturbance values owing to the presence of the magnetic modification of Fe_2O_3 as well as of magnetite.--W. Ayvazoglou.

COST OF MAGNETOMETER SURVEYING

By Dart Wantland

The Colorado School of Mines Magazine, vol. 20, No. 10, 1930, p. 24

Figures on the cost of magnetometer surveying, based on data furnished by an oil company, are given.

Four cases of surveying carried out in California, Western Canada, southwestern Texas, and Texas (Panhandle) are examined.

The following factors are to be considered: Weather, personnel, topography, transportation, location in respect to section corners, lost time, depreciation of car and instrument, and salary.

According to the author the personnel factor is by far the most important one in a discussion of real cost; that is, cost measured in quality of the work done.--W. Ayvazoglou.

PRELIMINARY RESULTS OF GEOLOGIC-PROSPECTING WORK CARRIED OUT BY THE GEOLOGICAL COMMITTEE IN 1929 IN THE REGIONS OF IRON-ORE DEPOSITS (IN RUSSIAN)

By A. J. Serk

Osvedomitelniy bulleten po Poleznim Iskopaemim, vol. 3, No. 1, 1930, pp. 6-13

The author describes in this article the geophysical work carried out in addition to the geologic survey and drilling in the following regions of the U.S.S.R.:

In the Krivoy Rog a systematic magnetic method of prospecting was used in the region of Melitopol and Krementchug. An anomalous strip about 70

kilometers in length was established to the S.SW. of the city of Orekhov, as well as several other magnetic anomalies in the region of Krementchug and Nikopol.

New places of magnetic anomalies were discovered in the Troitsko-Osamsk deposits, as well as in the regions of Minusinsk and Nerchinsk in Siberia.--W. Ayvazoglou.

3. SEISMIC METHODS

UN NOUVEAU TYPE DE SISMOGRAPHE PHOTOGRAPHIQUE

(A NEW TYPE OF PHOTOGRAPHIC SEISMOGRAPH)

By P. Guido Alfani

Ciel et Terre, vol. 46, No. 6, 1930, pp. 147-154

The author of this article introduced at the end of 1925 at the Ximenien Observatory a photographic seismograph of Galitzine type but with some constructive changes. The new apparatus was tested during a long period of time and the details of its construction are published now after the results of tests have proved to be very satisfactory.

In this article Alfani gives the description of the seismograph (including the registering apparatus and the lamp) as well as its photographic picture and schematical design. In conclusion he says that, although only the well-known principles are utilized, the apparatus differs from those used generally in the following important particulars: (1) A new method of connection is used between the pendulum and the amplifying system by which the mechanical friction is omitted entirely, and (2) the coefficients of amplification and damping are much higher than those obtained in the other modern seismographs.

The value of the apparatus is increased still more owing to its simple construction, low price, facility in operation and perfection of seismograms. Two seismograms taken at the same time, one by a Galitzin seismograph and the other by the new photographic seismograph, are added to the article for comparison.--W. Ayvazoglou.

NÄHERUNGSFÖRMELN ZUR BERECHNUNG DER AMPLITUDEN DER ELASTISCHEN WELLEN DIE BEIM DURCHGANG EINER GEGEBENEN WELLE DURCH EINE UNSTÄTIGKEITSFLÄCHE ENTSTEHEN

(APPROXIMATE FORMULAS FOR THE CALCULATION OF THE AMPLITUDES OF ELASTIC WAVES GENERATED AT THE PASSAGE OF A GIVEN WAVE THROUGH A LAYER OF DISCONTINUITY)

By H. P. Berlage, Jr.

Gerlands Beiträge zue Geophysik, vol. 26, No. 2, 1930, pp. 131-140

K. Zoeppritz in his article (Ueber Reflexion und Durchgang Seismischer Wellen durch Unstetigkeitsflächen, Nachrichten Göttingen, 1919, p. 66) has

worked out the extremely complicated relations between the amplitudes of the longitudinal and transversal reflected and refracted waves, generated at the passage of longitudinal and transversal seismic waves through a layer of discontinuity, in the abridged form of matrices.

The formulas given in this article are easier to operate with, but claim to be exact only in case of:

1. The reflection by a free surface (density of the second medium $= 0$);

2. Identity of the two media;

3. Reflection by a solid boundary (density of the second medium $= \infty$);

4. Any given density of the media, if the angles of incidence of the primary wave are 0 or 90°.

In every other case the formulas yield approximate values for the amplitudes of the secondary waves, with an error of probably not more than 0.1 of the amplitude of the incident wave.

The ratio of the velocities of propagation of condensational and distortional waves has been supposed to amount to 3. Thus liquid media have been ruled out.

The formulas should be used only in cases when the denser medium possesses greater velocity of propagation. Moreover, they do not extend to cases of total reflection.

The solution of two practical problems closes the paper.--Author's abstract.

DIE ERSTE BEWEGUNG BEI EINEM ERDBEBEN

(THE FIRST MOVEMENT PRODUCED BY AN EARTHQUAKE)

By M. Hasegawa

Gerlands Beiträge zur Geophysik, vol. 27, No. 1, 1930, pp. 102-125

Based on the quadrant distribution of the pull and shock of the first movement produced by an earthquake, as calculated by T. Shida, it is possible to draw a supposition concerning the break inside of the material; this may be formulated as follows: The internal break is caused by a shearing pressure in a plane; the breaking movement depends only on this shearing pressure independent of other internal pressures existing there. The theoretical distribution of the first movement of P and S caused by this shearing at one point of the break is calculated. The probable direction of the propagation of the break in the hypocenter is at the same time determined by this mathematical calculation. The distribution of the first movement depends on the proceeding of the break. If the

breaks have the same direction in all the points in the region of the hypocenter, then the first movement is distributed, in the first approximation, equally to that at the point of the break, although the amplitude ratio is more or less deranged.

It was proved by observation of the Tango earthquake that the pull and shock of the first movement produced by the break in a certain azimuth corresponds to the pull and shock of the first P-wave at the distant stations of the same azimuth. From this relation a theoretical conclusion is drawn, that the amplitude and the period of time of the first movement in the earthquake diagram are not the decisive factors for the first movement of the break but that they depend directly on the proceeding of the break.

The earthquake break (with the exception of comparatively deep earthquakes) has, as it seems, a tendency to be distributed in case of a horizontal shearing pressure along a vertical plane. The author examines also the distribution of the first movement in case of an inclined orientation of the break.

In conclusion, the author draws attention to some problems concerning the quakes caused by collapses and explosions.--Author's abstract translated by W. Ayvazoglou.

AN ANALYSIS OF THE S-WAVE

By Frank Neumann

Bulletin of the Seismological Society of America,
vol. 20, No. 1, 1930, pp. 15-33.

Of the three elementary wave types radiating from a seismic focus (1) compressional or P-waves, (2) transverse or S-waves, and (3) surface or L-waves) the author examines in this article the S-wave, the nature of which remains to be explained in such a manner that it can be recognized from the motion of the earth particle at a seismographic station.

Headings of the article:

1. Description of the S-wave group;
2. Displacement diagrams and discussion of errors;
3. Illustrations;
4. Preliminary results;
5. Supplemental note on the theory of the S-wave.

In the summary to the article the author says:

The following empirical law governing the azimuth of S-wave impulses is advanced: (1) The azimuth of the horizontal S-wave impulse at the focus (the horizontal component of the initial earthquake displacement)

is equal to (2) the azimuth from station to epicenter, plus (3) the azimuth from epicenter to station, or back azimuth, minus (4) the azimuth of the S-wave displacement at any station; all azimuths being counted from north around by east through 360°. It states that a common direction of horizontal impulse or thrust at a seismic focus can be calculated from S-wave azimuths observed at two or more distant stations. These focal thrusts appear to be either parallel or normal to fault lines.

A theory is later developed which appears to substantiate the above-named law in so far as it concerns simple horizontal focal displacements. S-wave impulses, due to a simple vertical displacement at a focus, apparently follow the laws of longitudinal motion, but arguments are advanced to question the existence of this type of activity. The displacements are longitudinal only in the sense that they occur in the planes of the rays; they are not compressional waves.

Three distinct types of activity are found in the S-wave group.--W. Ayvazoglou.

SEISMIC METHOD OF PROSPECTING (IN RUSSIAN)

By A. Seleznev

Vestnik Komiteta po delam izobreteniy (Gazette of the Committee for Examination of Inventions), No. 5 (67), May, 1930, pp. 1-3

The author discusses briefly the seismic method of prospecting and the use of seismographs and geophones.--W. Ayvazoglou.

4. ELECTRICAL METHODS

GEOELECTRICAL EXPLORATION METHODS USED IN OIL FIELDS

By Helmer Hedstrom

The Oil Weekly, vol. 58, Nos. 6 and 8, pp. 34-37 and 32-34.

The first part of this article presents, in a simple and nontechnical way, the principles underlying the use of geoelectrical methods for mapping of geologic structures. Different ways of applying electrical measurements for the surveying of structures are then described, and it has been attempted to make this presentation as elementary and easily understood to the average reader as possible. Detailed technical descriptions, which would be of interest mainly to those versed in the art of geoelectrical exploration, have been omitted, and reference to everyday physical phenomena are used to make clear the electrical phenomena in the subsurface. Thus the behavior of an electric current passing through the bordering surfaces between parallel layers with different conductivity is compared to the refraction and reflection of light at the contact planes between media like air, glass, and water (this comparison is, of course, fully justified, since the equipotential surfaces of an

electric current in such "stratified conductor problems" behave exactly like the wave fronts of light entering a stratified system). Further, the electromagnetic induction of electric currents in subsurface beds is explained by reference to the action of the industrial transformer and to simple laboratory experiments.

Of the geoelectrical methods dealt with, the electromagnetic process is stated to be the most important and most widely used. This method, which is given a fairly complete description, consists in measuring the different phase components of the electromagnetic field produced by an alternating current in a wire laid out on the ground. The measuring apparatus, as employed in the field work, is shown in a photograph; it consists of a search coil and a compensator, referred to in the article as "an instrument used in physical laboratories." The electromagnetic field is measured in the semiabsolute units of microgauss per ampere primary current.

It is explained, and shown by a diagram, how the presence of conducting beds in the subsurface will modify the alternating electromagnetic field, and how it is possible, by the measuring of this field, to determine not only the depth of such "electrical key beds," but also their electrical characteristics.

The result of a survey by this method is a structure map showing contour lines drawn on some outstanding "electrical key bed," with a depth ranging from 200 to 1,500 feet. It is stated that this process is much faster and cheaper than core drilling, for which it offers a substitute.

The second part of the article presents some examples of practical results obtained by the electromagnetic method. Such investigations have been used for a variety of structural problems in Texas, Louisiana, and California, where up to now a total area of between 1,500 and 2,000 square miles has been covered by electromagnetic surveys. Four maps and one diagram with sections are given, which show good agreement between the results from the electrical survey and the geological data obtained from subsequent drilling.--Author's abstract.

ZUR MESSUNG DER ELEKTRISCHEN LEITFÄHIGKEIT DER ERDE DURCH INDUKTION

(ON THE MEASUREMENT OF THE ELECTRICAL CONDUCTIVITY OF THE EARTH BY INDUCTION)

By J. Koenigsberger

Physikalische Zeitschrift, vol. 31, No. 10, 1930, pp. 478-485

The vertical component of the magnetic field strength produced by induced currents is calculated. These currents were induced in an infinite conducting semispace (Halbraum) by means of a circular current with a fixed radius passing through a very thin wire placed at the border of the semispace. Skin effect, small phase displacements, and screening off (Abschirmung) were neglected in the beginning. The errors produced by this were estimated. The semiempirical formula for the magnetic field of a current in a flat wire-circle is discussed and proved by observations. Only the first part of this formula is used during the integration; errors occurring by this are estimated.

The theory is applied to the earth, considering it a semispace; the observations proved that the resistance of the upper earth strata agreed in order with the resistance expected (3.10^4 ohms per cubic cm.).

For penetrating to depths of over 1 to 20 kilometers, frequencies less than 500 H are necessary. Sources of experimental errors and difficulties caused by this are explained briefly.--Author's abstract translated by W. Ayvazoglou.

THE INFLUENCE OF RAIN ON THE ATMOSPHERIC ELECTRIC FIELD

By A. Venkata Rao Telang

Terrestrial Magnetism and Atmospheric Electricity, vol. 35, No. 3, 1930, pp. 125-131

Photographic records of the atmospheric-electric field variations at times of rains are examined along with pluviograms covering these periods. It is found that the normal effect of rain is to reverse abruptly the normal positive field in the early stages, while in a few cases it results in a sudden reduction without change of sign. This abrupt change disappears speedily when the disturbing shower is of short duration. An analysis of the results of G. C. Simpson at Simla (1908-1909) in regard to the charge brought down by rain is given. It is shown that in the earliest stages of charged rain, in more than 85 per cent of the cases the net charge brought down by rain is positive. It is then shown that the effect of rain on the field is due to the transfer of charge by the rain from the upper air.

Calculation shows that the charge transferred by the rain is of the order necessary to produce the observed changes of potential. The explanation covers the cases of heavy rain attended with frequent reversals and very high values of the field.--Author's abstract.

DIE HERTZSCHEN GLEICHUNGEN UND DEREN LÖSUNG FÜR DAS ÄUSSERE ERDMAGNETISCHE FELD

(HERTZ'S EQUATIONS AND THEIR SOLUTION FOR THE EXTERNAL EARTHMAGNETIC FIELD)

By A. Sloutschanowsky

Gerlands Beiträge zur Geophysik, vol. 26, No. 3, 1930, pp. 333-350

The subject of the proposed work consists of finding out the solutions of Maxwell Hertz's equations under the following conditions: A homogeneous medium rotates about the axis R with a small constant angular velocity W_0 under supposition that the axis R in its turn rotates with a constant angular velocity W_0 about an immobile axis R_0 to which R is parallel. The region to be considered is the space between the two concentric spheres with their center at the point O which is taken on the axis R and remains on it during its motion together with the medium. The components of the magnetic field, which alone are subject to our consideration, are reported to the system of the coordinates with the center

at the point O, firmly adjoined to the medium, which system in such manner wholly participates in the motion of the considered medium. The indicated solution can, to a certain degree, serve for the representation of the external magnetic field of the earth from its surface to the higher layers of the atmosphere.--Author's abstract.

SUR LES PROSPECTIONS ÉLECTRIQUES DU PÉTROLE

(ON ELECTRICAL PROSPECTING FOR OIL)

By A. Belluigi

La Revue Pétrolifière, No. 394, October, 1930, pp. 1451-1454.

Based on experience gained with electrical methods of prospecting, two opposite opinions on the direct discovery of oil are maintained at the present time; one supports the possibility of the direct discovery and the other denies this possibility.

In this article Belluigi discusses this question and defends his viewpoint, expressed in one of his previous works (Electrical Prospecting for Oil, L'Elèttricita, No. 4, 1928) that all the different systems of electrical prospecting for oil may give good results if applied in places indicated and designated by geologists.--W. Ayvazoglou.

THE EARTH RESISTIVITY METHOD OF GEOPHYSICAL PROSPECTING; SOME THEORETICAL CONSIDERATIONS

By G. F. Tagg

The Mining Magazine, London, vol. 43, No. 3, 1930, pp. 150-158

Theoretical considerations illustrated by a series of curves and an actual example are given.

In conclusion the authors says that the extent to which results obtained theoretically can be of assistance in translating the observations obtained in the field must, of course, depend upon how closely the actual formations investigated approximate the conditions assumed in the theorctical investigations. In cases where the practical conditions approximate fairly closely those which have been considered, the results of the theoretical treatment given should serve as a useful guide to the way in which the practical observations are to be interpreted.--W. Ayvazoglou.

ELECTRICAL LOGS AND CORRELATIONS IN DRILL HOLES

By Conrad and Marcel Schlumberger

Mining and Metallurgy, vol. 10, No. 275, 1929, pp. 515-518

By making an "electrical log," the authors mean an operation to determine electrically the characters of formations traversed in drilling, the

determination being carried out by means of measuring the electrical resistivities of the rocks along the uncased portions of drill holes. As in the case of an ordinary core record, this process indicates a geological study of the strata, and for that reason it is given a similar designation.

The paper is divided into three parts:

1. Electrical resistivities of rock formations: In this section the authors discuss in considerable detail the factors affecting the resistivities of various kinds of rocks.

2. Technique employed in making these electrical logs: In order to measure the resistivity of a formation at a certain depth in a drill hole, appropriate apparatus is lowered on the end of an insulated electric cable to the given horizon. The measuring arrangement is especially designed to eliminate entirely the perturbations due to the conductivity of the water in the drill hole and to the phenomena of polarization provoked by the current flow. The data obtained permit drawing up a diagram which constitutes the Schlumberger electrical log. The illustrations show several such logs.

3. Applications of the electrical log: The recovery of a core and its complete lithologic and paleontologic examination will, of course, almost always give more information than will the determination of the electrical resistivity of the rock to which it belongs. Thus, the electrical "core" should be considered as an auxiliary to the core of stone and is called on to replace the latter simply because it is much cheaper, more rapid, and frequently more easily employed.--W. Ayvazoglou.

A NICKEL-COPPER DEPOSIT IN NEW BRUNSWICK, CANADA

By Bela Low

Engineering and Mining Journal, vol. 130, No. 3, 1930, pp. 115-118.

The article begins with a brief description of the geology of the region. An electrical survey on the Rodgers, Hall-Carroll, and intervening ground was carried out before any drilling was done. The inductive method was applied. The results of trenching checked well with the electrical conductor outlined in the survey. On the Rodgers lens a strong indication of the presence of a conductor coincided well with the known sulphide lens. Owing to the fact that outcrops were scarce, the area being partly farm land and partly timber of second growth, it was deemed best to establish favorable locations for additional explorations by means of a magnetic survey.

Regions of magnetic anomalies were found. No exploration work has as yet been done along these magnetic anomalies, but the logical assumption is that they represent pyrrhotite bodies.

Maps are given showing (1) schist-gabbro contact and the location of two sulphide lenses and magnetic anomalies, (2) trenches and diamond-drill holes on Rodgers lens, and (3) dip-needle readings and position of electrical conductor in relation to surface outline of Rodgers lens are given.

Results of metallurgical research conclude the article.--W. Ayvazoglou.

GEOPHYSICS USED IN TRACING KEWEENAWAN FAULT

Editorial note

Engineering and Mining Journal, vol. 130, No. 5, 1930, p. 245

The geophysics department of the Michigan College of Mining and Technology has undertaken the important geological task of locating definitely the Keweenawan fault in localities where its real position has never been established.

The Keweenawan is the major fault of the district, bounding the copper-bearing series on the south from the end of the Keweenaw peninsula to Lake Gogebic.

The character of the fault over its entire width can be determined, and the depth of the rock almost accurately calculated, by means of the varying electrical resistivity of the formations.--W. Ayvazoglou.

5. RADIOACTIVE METHODS

EIN VERBESSERTES VERFAHREN ZUR BESTIMMUNG DES GEHALTES DER FREILUFT AN RADIUMEMANATION

(AN IMPROVED PROCESS FOR THE DETERMINATION OF THE CONTENT OF RADIUM EMANATION IN FREE AIR)

By Walter Kosmath

Gerlands Beiträge zur Geophysik, vol. 27, No. 1, 1930, pp. 42-46

In this article the author shows that the cocoanut-charcoal process (the "suction method"), although improved in 1928, is not entirely suitable for the determination of the content of RaEm at a certain moment, or for the determination of the vertical distribution of the emanation.

The author mentions the demands to which a method for the determination of the content of RaEm at a certain moment may answer.

The method suitable to these demands is discussed and its advantage in comparison with the "suction method" is noticed.

The process examined in the article may be of importance also in connection with bioclimatic, balneologic, as well as medical questions.

The proceeding can be employed also during flights in airplanes and balloons.--Author's abstract translated by W. Ayvazoglou.

UNTERSUCHUNGEN ÜBER SCHWERE JONEN IN DER ATMOSPHÄRE

(INVESTIGATIONS ON HEAVY IONS IN THE ATMOSPHERE)

By Hans Israel

Gerlands Beiträge zur Geophysik, vol. 26, No. 3, 1930, pp. 283-313

With the apparatus described in the "first report" (Gerlands Beiträge, vol. 23, 1929, p. 144) measurements were made twice a day during one year and the number of small and large, positive and negative ions, as well as the nuclei of condensation, were counted, the latter with Aitken's Pocket Dust-Counter. A statistical study of the material obtained was made in connection with the other meteorological data. The variation of the number of large and small ions was always an opposite one. The nuclei of condensation had shown, of course qualitatively only, an attitude similar to that of the large ions. The influence of the large city upon the number of ions was very great. Their dependence on the air body could not be established beyond doubt owing to the disturbances caused by the city. Their relation to the relative humidity had shown that with the increase of humidity the conditions for the origination of large ions were more favorable. The proportion between the nuclei and large ions was not constant and changed with the meteorological conditions. The distinct annual and daily period of the number of ions seems to be in conformity with the amount of the inter-atmospherical change (Massenaustausch in der Atmosphere).--Author's abstract translated by W. Ayvazoglou.

GEOPHYSICAL PROSPECTING - RADIOACTIVE METHODS

By John H. Wilson

The Colorado School of Mines, vol. 19, No. 6, 1929, pp. 19-21

A brief description of the properties of the radioactive materials and tables of radioactive elements are given. The author discusses the following methods of radioactive prospecting:

1. Sample examination.

2. Atmospheric electricity method.

3. Investigation of soil gases.

4. Penetrating radiation method.

Five figures and a selected bibliography are added.--W. Ayvazoglou.

6. GEOTHERMAL METHODS

OBSERVATIONS GÉOTHERMIQUES DANS LE FORAGE STEBNIK 1

(GEOTHERMAL OBSERVATIONS IN THE STEBNIK 1 WELL)

By Stanislaw Zych

Institut de Géophysique et de Météorologie de l'Université de Lwow,
vol. 4, Communication No. 44, 1929, pp. 844-848

Data of temperatures measured on December 2, 1927, at different depths in the Stebnik 1 well (near Boryslaw) are given in a table.

The increase of temperatures with the depth is small and distributed regularly. For the differences of depth of 50 meters the increase varies between 0.75 and 0.85° C. The mean geothermal degree is 63.4 meters.

By comparing this value with those obtained in Boryslaw and Kalusz for the same differences in depth, shown in another table, it was established that the depth temperatures in the Stebnik well are considerably lower. This seems to be probably in relation with the salt layers and potassium salt layers crossed by the wells at a depth from 116 to 859 meters. In Kalusz the smallest increase was also observed in the salt layers--that is between 174 and 261 meters of depth.--W. Ayvazoglou.

DIE VORAUSBESTIMMUNG DER GEBIRGSTEMPERATUR IM INNERN VON GEBIRGSMASSIVEN

(PREDETERMINATION OF TEMPERATURE OF ROCKS INSIDE OF MOUNTAIN-MASSIFS)

By K. Pressel

Zeitschrift des Vereins deutscher Ingenieure,
vol. 73, No. 5, 1929, pp. 162-164

The author discusses the important question of determining the temperature of rocks inside of mountains. An attempt was made to solve this problem by mathematical calculations. Pressel describes methods in which the experiments are carried out on models. Possibilities of thermal and caloric model experiments are mentioned briefly. A third model experiment, an "electric" one, is described in more detail. Its usefulness was proved by observations made in two well-known tunnels (Gotthard, Simplon). The results of the latter examination are given.

The following items are discussed: (1) Importance of temperature determination; (2) literature on temperature determination; (3) influences upon the temperature inside of mountains; (4) thermal model-experiment; (5) caloric model-experiment; (6) electrical model-experiment.--W. Ayvazoglou.

GEOPHYSICAL PROSPECTING - GEOTHERMAL METHODS

By John H. Wilson

The Colorado School of Mines Magazine, vol. 19, No. 8, 1929, pp. 13-17

Owing to the fact that geothermal methods may give valuable information under certain conditions it can be expected that future improvements in technique and in interpretation will provide a greater application in certain types of prospecting problems.

In this article the author explains the definitions and terms used in connection with geothermal research work and gives a table of thermal conductivities for a number of substances.

Under the heading "The heat of the earth" Wilson enumerates the conditions due to which the actual isogeothermal surfaces depart from the ideal shape of oblate spheroids as calculated theoretically. These conditions are given as follows: (1) Ocean floors, (2) mountain ranges and valleys, (3) nature of the rocks, (4) dip of the strata, (5) structural uplift and movement, (6) chemical action, (7) volcanic activity, (8) underground circulation of waters, and (9) climatic changes.

A number of different types of thermometers and cases used for depth-temperature measurements in wells are shown.

Interpretation of depth-temperature curves, illustrated by graphs, is discussed.

A figure of Salt Creek oil field, Wyoming, showing structural contours on Second Wall Creek sand and depths to the isogeothermal surface of 80° (according to Van Orstrand) is given, being one of the best examples of the relation between structural uplifts and geothermal anomalies.

In conclusion some other applications of the depth-temperature measurements are mentioned.--W. Ayvazoglou.

GEOTHERMAL VARIATIONS IN OIL FIELDS OF LOS ANGELES, CALIFORNIA

By Anders J. Carlson

The Bulletin of the American Association of Petroleum Geologists,
vol. 14, No. 8, 1930, pp. 997-1011

Data obtained from temperature measurements in oil wells of the Los Angeles basin are presented, and some causes of abnormal temperature conditions and factors affecting the interpretation of geothermal data are noted. The relations of geothermal gradients and isothermal depths to structure are discussed for the Santa Fe Springs, Long Beach, and Torrance fields. Variations in the geothermal constants are considered with respect to the geology of the region, and the possible correlative value of temperature data is suggested.--Author's abstract.

7. UNCLASSIFIED METHODS

TRANSACTIONS OF THE AMERICAN GEOPHYSICAL UNION

Tenth Annual Meeting, April 25 and 26, 1929

Eleventh Annual Meeting, May 1 and 2, 1930

National Research Council, June, 1930, Washington, D. C., 314 pp.

In the introduction to this volume the general secretary of the union, Dr. J. A. Fleming, gives the following brief summary of the papers presented at the meetings:

> The tenth general assembly was held jointly, following a short business session of the Union, with the Section of Oceanography to hear general-interest papers which had been prepared for the program of the Section of Oceanography in addition to a number presented the preceding day at the Section meeting. The Section of Geodesy discussed progress-reports on gravity work in Mexico and in Europe and America and on geodetic work in Canada and in the United States, and papers on recent developments of geodetic methods, instruments and computations. In the Section of Seismology the papers were on surface-waves, the seismicity of the arctic, the forces and movements at the earthquake origin, and the practical application of earth-vibrations from dynamite blasts. The Section of Meteorology discussed the report of the meteorological division of the Committee on the Physics of the Earth of the Division of Physical Sciences of the National Research Council. The Section of Terrestrial Magnetism and Electricity held a symposium on physical theories of magnetic and electric phenomena. The Section of Volcanology heard communications on volcanic oceanic islands and the volcanic history of the San Juan Mountains of Colorado.
>
> At the eleventh general assembly, in addition to a series of six papers on "The utility of geophysics" in geodesy, seismology, meteorology, terrestrial magnetism and electricity, oceanography, and volcanology, the Union was addressed by Sir Hubert Wilkins on "The advisability of geophysical investigation in the arctic by submarine." Progress reports on work in Canada, Mexico, and the United States, on methods for the precise measurement of time and determination of gravity, and on correlative studies of latitude-variations featured the meeting of the Section of Geodesy. In view of common interests, the Section of Seismology, after a short business session May 1, met jointly with the Eastern Section of the Seismological Society of America on the morning and afternoon of May 5 at the Bureau of Standards and May 6 at the Georgetown University; six of the papers were presented by members of the Union and a symposium on the publication of earthquake data was led by Harry Fielding Reid of the Union. The Section of Meteorology discussed the relation between the Section of Meteorology of the International Geodetic and Geophysical Union and the International Meteorological Committee, as well as reports on

units of time, solar radiation, details of instruments, data required for indicating the climatological character of a region, and final report on the Bulletin on Meteorclogy in the series "Physics of the Earth" to be published by the National Research Council. The papers given before the Section of Terrestrial Magnetism and Electricity dealt with the proposed International Polar Year 1932-33, the significance and importance of continuance of magnetic and atmospheric-electric observations on the oceans with accounts of instruments suitable for work at sea, secular variations in the United States and over the earth's surface, and possible relationship of earth-movements and terrestrial-magnetic variation and of the aurora and the earth's magnetism. The Section of Oceanography had two sessions, one May 1 and a second May 2, hearing papers concerned with oceanographic work in its various phases and particularly with relation to physical oceanography and its relations to marine biology and meteorology. The program of the Section of Volcanology dealt with volcanic activity and the central African volcanoes.

The Union was favored at both the tenth and eleventh annual meetings by papers from members of the national committees of Canada and Mexico reporting on the progress of governmental activities and on results of geophysical researches stimulated by the committees. The active spirit of international cooperation on the part of Canada, Mexico, and the United States was again well evidenced in these papers and by the attendance of many visitors during the two annual assemblies.--J. A. Fleming.

GULF COAST RICH IN OIL

By T. E. Dabney

Manufacturers Record, vol. 98, No. 6, August, 1930, p. 56

In this article the author briefly summarizes the results obtained in Southern Louisiana and Texas by geophysical methods of prospecting (110 discoveries made by geophysical methods and 48 by nongeophysical methods).

The principles of the work of the torsion balance, seismograph, magnetometer, and the gravity pendulum are mentioned.

According to Dabney, since 1921, nearly $7,000,000 has been spent by oil companies on geophysical methods on the Gulf Coast alone, and at present about $300,000 a month is being spent in this region.

The Gulf Coast must be considered one of the largest potential petroleum reserves known to the industry.--W. Ayvazoglou.

NOTE ON A RECENT ARTICLE BY DR. HOPFNER

By Walter D. Lambert

Gerlands Beiträge zur Geophysik, vol. 26, No. 2, 1930, pp. 182-184

The article mentioned is entitled "Zur Bestimmung der Erdgestalt nach isostatischen Gesichtspunkten" (see Geophys. Abs. No. 13, p. 20).

Lambert's abstract of the present article reads as follows:

The various determinations of the figure of the earth published by Hayford are not all strictly comparable, and indeed they are not intended to be so. This fact seems to have given rise to misunderstandings. To remove these some brief explanations are given. The effect of isostatic methods of reduction on the figure of the earth and value of various astronomical methods of determining it are briefly discussed.--Author's abstract.

UEBER DIE KONVERGENZ DER REIHE FÜR DAS ÄUSSERE RAUMPOTENTIAL

(ON THE CONVERGENCE OF THE SERIES FOR THE EXTERIOR POTENTIAL OF SPACE)

By F. Hopfner

Gerlands Beiträge zur Geophysik, vol. 27, No. 1, 1930, pp. 36-41

The question concerning the sphere of convergence in the development of the series for the exterior potential of a sphere is especially important for geophysics owing to the fact that the validity of a number of fundamental formulas, such as Clairaut's theorem, Bruns' theorem and others, depend on the answer to this question.

In this article the author shows that the development of the potential of space in series of spherical harmonics for the exterior space is convergent also in a certain region of the interior space of the attracting mass.--W. Ayvazoglou.

THE GEOLOGIST OF TODAY AND YESTERDAY

Editorial note

The Petroleum Times, vol. 24, No. 610, 1930, pp. 451-454

After the description of the work of a geologist for mapping out the structure of the underground from surface observations the geophysical means of prospecting, especially for oil, are discussed. Seismic, gravitational, and magnetic methods are briefly examined. Cooperation of the geologist, geophysicist, and the palaeontologist is at the present time required in order to reduce the chances of failure in drilling for oil.--W. Ayvazoglou.

THE WORK OF THE GEOPHYSICAL INSTITUTE OF THE CENTRAL GEOLOGICAL AND PROSPECTING SERVICE IN U.S.S.R. IN 1928-29 (IN RUSSIAN)

By J. Lepeshinsky and V. Chernobrovin

Osvedomitelniy bulletin po poleznim iskopaemin, vol. 3, No. 2, pp. 15-19 and No. 3, pp. 18-23

The importance of geophysical methods of prospecting caused the creation in 1929 of a special Geophysical Institute for geologic-prospecting work with the following sections: Electrotechnical, magnetic, gravimetric, radiometric, seismic, and geothermal.

In this article the authors give the results of the work carried out during 1928-29 by the above-mentioned sections in the different regions of the U.S.S.R.--W. Ayvazoglou.

GEOPHYSICAL METHODS APPLIED TO EXPLORATION AND GEOLOGIC MAPPING

By T. M. Broderick and C. D. Hohl

U. S. Geological Survey, Professional Paper 144, 1929, pp. 156-168

The article forms a part of the work entitled "The Copper Deposits of Michigan," by B. S. Butler and W. S. Burbank. The authors discuss the possibility of application of geophysical methods to the exploration and geologic mapping of the region.

Theory and practice of magnetic surveying is mentioned especially.--W. Ayvazoglou.

INSTRUMENT FOR DETECTING METALLIC BODIES

By Theordore Theodorsen

Journal of the Franklin Institute, vol. 210, No. 3, 1930, pp. 311-326

This paper gives a description of a new instrument recently developed by the National Advisory Committee for Aeronautics at the Langley Memorial Aeronautical Laboratory (N.A.C.A. detector). The instrument was made for the immediate purpose of locating unexploded bombs which were known to have been dropped from airplanes at targets in close proximity of the site of the new Seaplane Towing Channel at Langley Field, Va. The new "detector" successfully located a number of bombs buried on and near the projected site. It is of a simple design and requires no skilled operators.

The author gives a brief theoretical survey of the general nature of the difficulties encountered in the design of sensitive detectors of this type. He points, in particular, to the importance of avoiding capacity and resistance effects, and outlines other essential factors contributing to the success of the new detector.

Thirteen figures illustrate the article.--Author's abstract.

FORTSCHRITTE DER ANGEWANDTEN GEOPHYSIK

(ATTAINMENTS OF APPLIED GEOPHYSICS)

By J. B. Ostermeier

Internationale Zeitschrift fuer Bohrtechnik, Erdölbergbau und Geologie, vol. 38, No. 16, 1930, 14 pp.

In this article, which is a reprint of a lecture delivered by Ostermeier before the second general meeting of the International Oil-Union held on June 14, 1930, in Vienna, the author notices the attainments made by geophysical methods of prospecting.

Gravitational, seismic, electrical, and magnetic methods are discussed, based on examples.

Photographic pictures of instruments and a series of diagrams are shown in 21 figures illustrating the article.--W. Ayvazoglou.

SOCIETY OF PETROLEUM GEOPHYSICISTS

Editorial note

Bulletin of the American Association of Petroleum Geologists, vol. 14, No. 10, 1930, pp. 1364-1365

Information on the organization of the Society of Petroleum Geophysicists is given. The object of the society is to bring together geophysicists who are engaged in oil work and to advance the science of the application of physics in oil geology.

The requirements for membership are enumerated.

The following officers were elected: President, Donald C. Barton; vice-president, E. E. Rosaire; secretary-treasurer, John Weinzierl.

Application blanks may be obtained from the secretary at 608 Petroleum Building, Houston, Tex.--W. Ayvazoglou.

8. GEOLOGY

SURFACE GEOLOGY OF COASTAL SOUTHEAST TEXAS

By Donald C. Barton

Bulletin of the American Association of Petroleum Geologists, vol. 14, No. 10, 1930, pp. 1301-1321

The formations at the surface in coastal southeast Texas are the "Lissie," Beaumont, Recent, and Recent Terrace deposits. The Hockley scarp, one of the most striking physiographic features of the region, is a flexure

scarp partly buried by alluvial deposits. The area in front of the Hockley scarp in southeast Texas is largely a deltaic plain composed of the coalescent deltas of late Pleistocene Trinity and Brazos Rivers. The sands of the area of the Lake Charles loams and sands, currently mapped as Lissie, are continuous with the sands of the distributary ridges and are contemporaneous with the surface Beaumont clay, and that part of the Lissie at the surface seaward of the Hockley scarp is younger than that lying landward of the scarp and is a sandy phase of the Beaumont. An uplift of 20 to 30 feet is indicated in Orange County and in Cameron and Calcasieu parishes. A veneer of Recent sediments covers the seaward edge of the Beaumont clay. Two, possibly three, post-Pleistocene terraces are present in the valleys of the major streams. The soils of the early Recent terrace show the introduction of material from the Permian "Red Beds."--Author's abstract.

DIE ENTSTEHUNG DER KONTINENTE

(THE ORIGIN OF THE CONTINENTS)

By Josef Geszti

Gerlands Beiträge zur Geophysik, vol. 27, No. 1, 1930, pp. 1-25

In the ancient form of the earth the liquid sima masses were everywhere,--in the same height of strata--enveloped by light, molten sial masses. This concentric distribution of material, however, has been disturbed later on. At present the continents consist of sial and the bottoms of the oceans mostly of sima materials (vertical separation of material). It has been shown that this distribution of masses was created by thermodynamic effect.

The elevation of the continents has been attributed to the condensation of the sima masses, specially on the point of becoming solid during the transition from liquid to crystallized state.

The origin of the thermodynamic effect is being attributed to the inhomogeneous sial masses containing various auxiliary mixtures.--Author's abstract.

USE OF AIRPLANE PHOTOGRAPHS IN GEOLOGIC MAPPING

By Walter A. English

Bulletin of the American Association of Petroleum Geologists, vol. 14, No. 8, 1930, pp. 1049-1058

The demand for airplane photographs during the World War caused a rapid development in equipment and methods. This demand and the improvements in airplane performance laid the foundation for the present commercial applications of airplane photography. Many oil companies use photographs in field mapping and find several peculiar features of the photographs of particular value in geologic mapping.

Mosaics of various types are used for compilation of data, but line maps and contour maps can be constructed from photographs if maps of a greater degree of accuracy are desired.--Author's abstract.

9. NEW BOOKS

Breyer, Johannes, Ueber die Elastizität von Gesteinen (On the Elasticity of Rocks). Published by the "Preussiche Geologische Landesanstalt," Berlin, 1929, 53 pp., 5 tables, 9 figures, 3 synoptical tables. Price, R.M. 3-75.

Castelfranchi, Gaetano, Engineer of the Ecole Superieure des Ingenieurs de Milan. Physique Moderne, Exposé Synthétique et Méthodique de la Physique d'Aujourd'hui et des travaux théoriques et experimentaux des plus grand physiciens contemporains (Modern physics, synthetical and methodical exposé of present physics and theoretical and experimental works of the most celebrated contemporary physicists). Translated into French by M. A. Quemper de Lanascol, Laureat de l'Institut. 660 pp., illustrations. 8-vo. Paris. Librairie Scientifique Albert Blanchard, 1930. Price, 70 fr.

Kohlrausch, Friedrich. Lehrbuch der praktischen Physik (Handbook of Practical Physics). 16th enlarged edition. Revised by W. Bothe, E. Brodhun, E. Giebe, E. Gruneisen, F. Hoffmann, K. Scheel and O. Schönrock. XXX and 800 pages, 395 figures. Leipzig, 1930. Teubner. Price: Stitched, R.M. 23, bound, R.M. 26.

Vernadsky, W. J. Geochemie in ausgewählten Kapiteln (Selected Chapters on Geochemistry). Translated from Russian by Dr. E. Kordes, Leipzig, 1930. Akademische Verlagsgesellschaft m.b.H. 370 pages. Price: R.M. 23; bound R.M. 25. Chapter I: History of geochemistry; II, Chemical elements and the different ways in which they occur in the earth's crust; III, Geochemistry of manganese and considerations of energy; IV, Silicon and silicates in the earth's crust; V, Carbon and the vital substance in the earth's crust; VI, The radioactive elements in the earth's crust.

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I. C. 6423 April, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

COMPENDIUM ON LIMES
IN HYDROMETALLURGY AND FLOTATION

This report presents the results of work done under a cooperative agreement between the U. S. Bureau of Mines and the Missouri School of Mines and Metallurgy, Rolla, Mo.

BY

R. G. O'MEARA, ALEXANDER M. GOW, AND W. T. SCHRENK

I.C. 6423
April, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

COMPENDIUM ON LIMES IN HYDROMETALLURGY AND FLOTATION[1]

- By -

R. G. O'Meara[2], Alexander M. Gow[3], and W. T. Schrenk[4]

INTRODUCTION

Lime is used extensively in the various industries. The National Lime Association[5] in an excellent and recent bulletin lists 23 important industries in which lime plays a vital role; and it thereby justifies the statement that "Lime is one of the most versatile and widely used reagents in the chemical and industrial world." There is, however, one application of this "universal" reagent which has not been mentioned. It is the use of lime in the flotation of ores. An appreciable percentage of the lime production is consumed by ore-dressing plants.

The uses of lime throughout all the industries are more or less related, but "lime" is a broad term; the varieties of lime are more numerous than the limestones from which it is made, and its use involves a complicated technique. The interrelated application and growing importance of lime in flotation warrant this additional publication on the various "limes," with emphasis on those uses related to the function of lime in flotation.

Lime in water treatment, cyaniding, and amalgamating will be mentioned, after which its use in flotation will be taken up.

This compendium was prepared by the Mississippi Valley Experiment Station of the United States Bureau of Mines and the Missouri School of Mines and Metallurgy, Rolla, Mo.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from the U. S. Bureau of Mines Information Circular 6423."

2 - Assistant metallurgist, U. S. Bureau of Mines, Mississippi Valley Experiment Station, Rolla, Mo.

3 - Assistant metallurgist, U. S. Bureau of Mines, Mississippi Valley Experiment Station, Rolla, Mo.

4 - Consulting chemist, U. S. Bureau of Mines, Mississippi Valley Experiment Station, Rolla, Mo., and Professor of Chemistry, Missouri School of Mines and Metallurgy, Rolla, Mo.

5 - National Lime Association, Lime: Its Uses and Value in the Industrial and Chemical Processes; Washington, D. C., 1930, 88 pp.

Lime was one of the first chemical substances used by mankind. Knibbs[6] gives an interesting historical sketch of lime, stating that it is as old as the use of fire, that it was used as an offensive weapon in war by throwing it into the faces of the enemy, and that magnesia was not clearly distinguished from lime until the eighteenth century:

> Lime was one of the first chemical substances to be used by mankind, and lime-burning is one of the oldest chemical industries. The process of calcining limestone is probably nearly as old as the use of fire, because in any limestone country the effect of lighting a fire on the stone would be to produce lime, but the intelligent application of the process must have come much later . . . . .
>
> According to Quietmeyer, lime, together with plaster of Paris, was used in Egypt about B. C. 2600, and by Soloman in Jerusalem circa 950 B. C., and it is probable that its use began at a much earlier age . . . . . . Xenophon (circa B. C. 350) records the wrecking near Marseilles of a ship carrying a cargo of linen and the lime required for its bleaching, but Cato, in B. C. 184, appears to have been the first to mention kilns for burning it. Dioscorides and Pliny (circa A. D. 75) both describe the production, slaking, and uses of lime . . . . . . Vitruvius, at the beginning of the Christian era, dealt at length with the use of lime for building purposes. Lime is mentioned in the Bible . . . .
>
> Little is known regarding the condition of the lime industry in medieval times, but a knowledge of its properties and its use for building purposes is reflected in the writings of the day . . . . .
>
> Quicklime would appear to have been used in the Middle Ages for offensive purposes in war, and the English hurled it in their enemies' faces at a naval battle just after the death of John in 1217 . . . . . . It was used by alchemists for causticizing the alkali metal carbonates in wood ashes and for other purposes, but it was so familiar a material that they seldom thought it worth mentioning, and no attempt seems to have been made to explain the nature of the process of calcination until the phlogistic period.
>
> The elucidation of the nature of the calcination process is due largely to Joseph Black, who showed that it consisted in the expulsion of 'fixed air.' . . . .
>
> Magnesia was not clearly distinguished from lime until the 18th century when Black showed that the former gave a soluble sulphate. It therefore has no individual early history. Magnesia alba appears to have come into commerce from Rome about A. D. 1700. The calcination of magnesium carbonate, like that of calcium carbonate, was explained by Black and Lavoisier.

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6 - Knibbs, N. V. S., Lime and Magnesia: Ernest Benn, Ltd., London, 1924, 300 pp.

Limestone, from which lime is made, is one of the important substances in the lithosphere. That it is widely distributed upon the earth is shown by Clarke.[7] He shows that the average limestone is only about 75 per cent pure and states as follows:

> . . . . . . Broadly, then, we may estimate that the lithosphere, within the limits assumed in this memoir, contains 95 per cent of igneous rock and 5 per cent of sedimentaries. If we assign 4.0 per cent to the shales, 0.75 per cent to the sandstones, and 0.25 per cent to the limestones we shall come as near the truth as is possible with the present data . . . .

Whether or not the figures are correct is of no importance to us other than it does show that the distribution is widespread. Nor has the United States been slighted in the matter of geologic distribution of limestone. The statistics published by the Department of Commerce, Bureau of Mines, for 1928[8] show that 41 of the 48 States have reported plants in operation for the burning of lime. In 1928 the four largest producing States were Ohio, Pennsylvania, Missouri, and West Virginia, in the order named. The total number of plants in operation in 1928 was 411. Keen competition existed among the producers and there was a tendency to overproduction and lower prices. If lime is the most plentiful alkali, it is also the cheapest. The average value, at the plants, of lime sold decreased from $8.75 a ton in 1927 to $8.18 a ton in 1928. In 1929 the average price for hydrated lime was $8.24 per ton. The average value for building lime was 85 cents less than in 1927, for chemical lime was 24 cents less, and for agricultural lime was 8 cents less. The production of lime in the United States from 1919 to 1929, inclusive, is given in Table 1, taken from Mineral Resources of the United States, 1928.[9]

The statistics given in Mineral Resources do not include a considerable amount of lime that is not a commercial product, but is burned for direct use in manufacturing, from stone either quarried or purchased by the manufacturer. This stone is reported to the Bureau as raw limestone and is included in the statistics on stone. It is used mainly by alkali works, sugar refineries, and smelters.

As mentioned in historical sketches on lime, the first recorded use was in mortar and plaster. The field has since extended so that lime is a necessity in many industries. Holmes[10] mentions the extensive uses of lime:

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7 - Clarke, Frank Wigglesworth, The Data of Geochemistry: U. S. Geol. Survey, Bull. 770, 1924, p. 34.

8 - Coons, A. T., Lime in 1928: Mineral Resources of the United States, 1928 - Part II, 1929, pp. 175-186.

9 - Coons, A. T., Work cited.

10 - Holmes, M. E., An Outline of the Uses of Lime: Chem. and Met. Eng., vol. 26, Feb. 15, 1922, pp. 294-300.

Table 1. - Lime sold by producers in the United States, 1919-1929

Year	Hydrated lime			Total lime		
	Number of plants in operation	Short tons	Value[1]	Number of plants in operation	Short tons	Value[1]
1919	93	779,408	$ 7,061,146	539	3,330,347	$ 29,448,553
1920	98	853,116	9,287,562	515	3,570,141	37,543,840
1921	105	792,970	7,421,637	520	2,532,153	24,895,370
1922	114	1,106,063	9,868,980	530	3,639,617	33,255,039
1923	121	1,225,928	12,229,598	467	4,076,243	39,993,652
1924	120	1,316,664	13,199,846	450	4,072,000	39,596,423
1925	134	1,560,848	15,287,461	450	4,580,823	42,609,141
1926	147	1,606,811	15,182,460	435	4,560,398	41,566,452
1927	157	1,596,906	14,581,695	417	4,414,932	38,638,413
1928	164	1,612,818	13,540,215	411	4,458,412	36,449,635
1929[2]		1,550,771	12,771,525	381	4,269,768	33,478,848

1 - The value given represents the value of bulk lime f. o. b. point of shipment and does not include cost of barrel or package.

2 - Advance figures from press release of Department of Commerce on "Sales of Lime in 1929;" Sept. 16, 1930.

The use of lime in mortar and plaster was, in past ages, its only important use. Coincident with our modern industrial development along technical lines, the lime industry has expanded enormously in the number of its technical uses and thereby has offered a fertile field for the activities of the investigator and writer . . . . . .

In the present article the author has approached the subject primarily from the point of view of a systematic arrangement in outline form of the uses of lime, grouping the various uses under headings indicating functions performed by lime. In order to show the unusually large number of functions that lime can perform, we have included not only the well-known uses that represent large tonnages but all the uses of which we have authentic records.

. . . .

In industrial technology it is convenient to divide the uses of lime into three major groups - viz., construction uses, agricultural uses, and chemical uses. The construction uses include those involved in building operations, the agricultural uses are limited to the uses on the farm, and the chemical uses include the very large number involved in manufacturing and technical processes. Although the allocation of certain uses is somewhat arbitrary, yet it seems to divide the field in accordance with the duties of the engineer, agronomist, and chemist.

Holmes gives in his article a table of the relative importance of the various uses of lime, compiled from data for 1921 given by the United States Geological Survey. The latest available figures (1928) taken from Coons[11] on lime are given in Table 2 and rearranged to have the same form as the table shown by Holmes.

The table shows that in 1928, 504,248 tons of lime was used for metallurgical work. The consumption of lime in flotation for the same period was about one-fifth of this amount. From this the conclusion is drawn that about 2.5 per cent of all the lime consumed is used in flotation.

CLASSIFICATION OF LIMES

Lime in the true sense of the word denotes calcium oxide, but within the industry, usage and the varied chemical composition of limestone, from which the lime is obtained by calcination, have extended the meaning so that it also connotes a large variety of products varying in composition with regard to the CaO--MgO ratio. A classification is given in Table 3 which is a modification of one by Knibbs.[12] This classification serves to remind the flotation operators that due attention should be given to the composition of the lime purchased.

Ladoo[13] discusses the divergence of commercial limes from the true chemical significance of the term lime as follows:

> . . . . . . By proper chemical nomenclature lime is calcium oxide (CaO), but, commercially, lime is understood to mean the product of complete calcination of a 'limestone,' which may vary from pure calcium carbonate ($CaCO_3$), to pure dolomite ($CaCO_3 \cdot MgCO_3$). Thus, commercial lime may contain as high as 45 per cent of magnesium oxide (MgO) as well as such impurities as silica, iron oxide, and alumina.

Another author[14] discusses the variation in limes in the following manner:

> The term 'limestone' is used to describe a class of rocks varying in composition from pure calcium carbonate to a mixture of 54.35 per cent calcium carbonate with 45.65 per cent magnesium carbonate. (According to definition by National Lime Association.) Any gradation between these limits may be found, and all limestones contain more or less impurities. In the same way lime may vary from pure calcium oxide to a mixture of calcium and magnesium oxides in the corresponding proportions.

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11 - Coons, A. T., Work cited.
12 - Knibbs, N. V. S., Work cited.
13 - Ladoo, Raymond B., Non-metallic Minerals; McGraw-Hill Book Co., New York, 1925, p. 310.
14 - Porter, J. M., Manufacture of Lime: Bureau of Standards Cir. 337, 1927, p. 4.

Table 2. - Relative importance of the various uses of lime

Uses	Percentage of total		Short tons	
	1927	1928	1927	1928
BUILDING	48.7	44.6	3,148,840	1,986,465
CHEMICAL:				
Sugar refineries	.4	.5	16,086	22,678
Tanneries	1.4	1.4	63,666	64,464
Glass works	1.8	1.7	78,994	76,161
Refractory lime (dead-burned dolomite)	8.5	10.1	374,415	448,761
Metallurgy	9.2	11.3	406,063	504,248
Paper mills	9.7	9.6	429,606	429,334
OTHER USES:	13.0	13.3		
Alcohol manufacturing and dehydration			2,199	2,302
Alkali works (ammonia, soda, potash)			19,610	6,675
Bleaching powder			5,723	14,970
Bleach liquid			10,308	18,954
Calcium acetate			9,618	1,643
Calcium carbide			27,037	32,596
Coke and gas manufacturing (gas purification and plant by-products)			29,988	31,011
Creameries and dairies			(1)	- -
Disinfectants (chloride of lime, etc.)			(1)	- -
Flour mills			(1)	- -
Glue			7,864	8,467
Insecticides (spraying material)			22,154	19,269
Oil and fat manufacturing			3,899	9,119
Oil refining			7,101	(1)
Paint (kalsomine, whitewash, varnish, etc.)			5,050	5,017
Rubber			4,192	3,361
Salt refining			1,210	2,423
Sand-lime and slag brick			43,064	47,527
Sanitation (sewage, garbage purification, etc.)			2,416	2,857
Silica brick			15,638	15,735
Soap			25,487	33,759
Textiles			1,128	1,460
Water purification			124,212	145,757
Wood distillation			11,682	3,899
Undistributed(2)			42,973	53,198
Unspecified			151,816	132,392
Agricultural	7.3	7.5	322,893	333,910
Total	100.0	100.0	4,414,932	4,458,412
Hydrated lime (included in above totals)	36.2	36.2	1,596,906	1,612,818

(1) Included in "undistributed."

(2) Lime used in acid neutralization and drying, acetic acid, asphalt and fertilizer filler, bichromate refining, buffing compounds, calcium phosphate, cellulose, corn products, creameries and dairies, cyanide manufacture disinfectants (chloride of lime, etc.), dyes, flour mills, food products, gelatin (edible), magnesia oil refining, oxygen, purification, retarder, and stock food.

Table 3. - Classification of limes

Oxide	Hydroxide	Composition of Carbonate		
		$CaCO_3$, per cent	$MgCO_3$, per cent	Other compounds, per cent
Calcium oxide (pure)	Calcium hydroxide (pure)	100	-	-
Lime High-calcium lime Quicklime Fat lime Caustic lime Shell lime	Hydrate of lime Hydrated lime Slaked lime High-calcium hydrated lime Caustic lime	90 to 100	0 to 10	0 to 10
Siliceous lime Argillaceous lime Hydraulic lime Lean lime	Hydrated siliceous lime, etc.	50 to 90	0 to 10	10 to 50
Magnesian lime Dolomitic lime	Magnesian hydrated lime; Hydrated magnesian lime, etc.	60 to 90	10 to 40	0 to 10
Dolomitic lime Dead-burned lime	Hydrated dolomitic lime	50 to 60	40 to 50	0 to 10
Dolomitic magnesia or dead-burned dolomitic magnesia	-	20 to 50	50 to 80	0 to 10
Magnesia Dead-burned magnesia	-	0 to 20	80 to 100	0 to 10

Eckels[15] gives the following classification of commercial limes:

Group A. High calcium limes: Limes containing less than 5 per cent of magnesia. The limes of this group differ among themselves according to the amount of silica, alumina, iron, etc., contained. A lime carrying less than 5 per cent of such impurities is a 'fat' or 'rich' lime, as distinguished from the more impure 'lean' or 'poor' limes.

15 - Eckels, Edwin C., Cements, Limes and Plasters: New York, 1928, pp. 98-9.

Group B. Magnesian limes: Limes containing over 5 per cent (usually 30 per cent or over) of magnesia. These limes are all slower slaking and cooler than the high-calcium limes of the preceding group, and they appear to make a stronger mortar. They are, however, less plastic or 'smooth', and in consequence are disliked by workmen. As commercially produced, they usually carry over 30 per cent magnesia.

Holmes and Fink[16] give a slightly different classification:

. . . . By the term commercial lime, we refer to the three and one-half million tons of product made yearly in this country by calcining limestone and which is placed on the market without any further processes of chemical purification. That this material may be quite different from corresponding c. p. chemicals is evident from the following definitions as adopted by the American Society for Testing Materials:

'Quicklime is a calcined material, the major part of which is calcium oxide or calcium oxide in natural association with a lesser amount of magnesium oxide capable of slaking with water.

'Hydrated lime is a dry powder made by treating quicklime with enough water to satisfy its chemical affinity under the condition of its hydration. It consists essentially of calcium hydroxide or a mixture of calcium hydroxide and magnesium oxide and magnesium hydroxide.'

Commercial limes are further classified by the A. S. T. M. on the basis of alkaline-earth oxide ratio as calcium and magnesium limes. In common practice calcium lime is one containing 85 per cent or more total CaO and magnesium lime is one containing more than 10 per cent MgO. In case the CaO content is very high, it may be called high calcium lime and in case the magnesium oxide is considerably more than 10 per cent it may be called high-magnesium lime.

Ladoo[17] gives practically the same classification with more information on the subject of impurities.

Lime is sold in two forms: (1) Quicklime (CaO), and (2) hydrated lime $Ca(OH)_2$. Hydrated lime is formed from quicklime by adding the requisite amount of water. When quicklime is exposed to the air for some time, it gradually absorbs carbon dioxide (CO_2) from the air and 'air-slakes,' or reverts to its original form of calcium carbonate ($CaCO_3$).

Commercial limes are classified according to their relative content of CaO and MgO as follows:

16 - Holmes, M. E., and Fink, G. J., Fundamental Physical and Chemical Properties of Commercial Lime; I - Available Lime Content: Chem. and Met. Eng., vol. 27, Aug. 23, 1922, p. 347.

17 - Ladoo, R. B., Work cited.

High-calcium lime: Not less than 90 per cent CaO.

Calcium lime: Not less than 85 per cent nor more than 90 per cent CaO.

Magnesian lime: Not less than 10 per cent nor more than 25 per cent MgO.

High magnesian lime: Not less than 25 per cent MgO.

Total impurities (exclusive of carbon dioxide) should not be over 5 per cent in selected lime or 7.5 per cent in run-of-kiln lime.

In the following[18], the leanness of limes is attributed to argillaceous components instead of magnesium oxides as is the general practice:

> Usually the limit of MgO for 'fat limes' is taken as 5 to 9 per cent. However, in Palermo and Lombardy they use fat limes obtained from dolomitic limestones which contain as high as 40 per cent MgO. These limes are free from or contain very little SiO_2, clay, etc., while these impurities are present in considerable quantities in 'lean' limes. Hence the 'leanness' of limes is due, not to MgO, but to SiO_2 or other argillaceous components.

Thus, as was said before, "lime" is a broad term. Rarely is commercial lime free from impurities, however, are not necessarily harmful; but designation should be made as to the type of lime under consideration. In general, limes may be classified as "high-calcium" or "high-magnesium," or as "fat" or "lean," depending on the chemical composition of the stone from which they were produced. Such classification is important since the different varieties have different properties. In addition to the composition, the manner of calcining also determines the character of the product.

MANUFACTURE OF LIME

The manufacture or burning of lime may be defined as the process of converting limestone into lime through the agency of heat. The impurities may or may not be removed. It is apparent, therefore, that the burning of lime is concerned merely with the removal of carbon dioxide from the stone. When the stone is heated to a sufficiently high temperature and kept there for a sufficiently long time, the carbonates are dissociated and the carbon dioxide is driven off as a gas.

In the manufacture of lime, limestone is quarried, broken into convenient sizes, and calcined in a kiln. Most of the quicklime produced in this country is made in shaft kilns in which the selected limestone is subjected to a temperature above that of the decomposition of calcium carbonate (900° C.) for a sufficient time to drive off the carbon dioxide. Commonly, kiln temperatures range from 1,200 to 1,400° C. The quality of limestone suitable for

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18 - Occhipinti, Francesco, Influence of Magnesia Upon the Character of Aerated Limes: Giorn. Chim. ind. Applicata, vol. 6, 1923, pp. 17-20.

burning may vary both chemically and physically within rather wide limits. Porter[19] says:

> . . . . It must be remembered that on account of the loss of about half the weight of the stone as carbon dioxide during the burning the proportion of every other constituent of the stone will be nearly doubled in the lime . . . . The amount of impurities (silica and oxides of iron and aluminum) permissible in a finishing lime (quicklime or hydrate) is very small, certainly not over 5 per cent (2-1/2 per cent in the stone).

The best burning of lime is obtained by using the minimum amount of heat. Porter[20] states as follows:

> There are three factors essential to the purpose of lime burning: (1) The stone must be heated to the temperature of dissociation of the carbonates; (2) the temperature must be maintained for a certain length of time; (3) the carbon dioxide evolved must be removed.
>
> Many experiments have been made to determine the temperature of dissociation of calcium carbonate. The best and most recent work done on this subject has been by Johnson and Mitchell. Both of these investigators found the dissociation temperature to be 898° C. The corresponding temperature for magnesium carbonate has not been determined as accurately and the results are not at all concordance, but the value 756° C. obtained by Mitchell in 1923 is probably quite correct. These figures are for a pressure of one atmosphere of carbon dioxide, a condition hardly ever met with in practice. The dissociation temperature of stones containing magnesium carbonate depends upon the manner of combination of the two carbonates. Some investigators have found the two materials to behave as separate compounds, dissociating at their respective temperatures. On the other hand, others have found the behavior to indicate the presence of a double salt, with the dissociation temperature lying between that of calcium carbonate and magnesium carbonate. Recent work conducted at this bureau indicated the latter condition to be true.
>
> After the dissociation temperature has been reached it must be maintained for some time in order to transfer the required amount of heat to the stone . . . .
>
> The physical properties of the stone undoubtedly have some influence on the amount of heat required and the time in which this heat can be transferred to any given temperature. Thus, it will take longer to burn a large piece of stone than a smaller one. A fine-grained, dense stone will conduct the heat more readily than

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19 - Porter, J. M., Page 6 of work cited.
20 - Ibid., p. 8.

one which is coarsely crystalline and porous. Practical experience seems to point to the fact that the crystals themselves require more heat to dissociate them than the more fine-grained stone. The time required to transfer a given amount of heat, roughly speaking, varies inversely with the temperature difference. Therefore, it has been found economical to use as high a temperature as possible in order to reduce the time required for burning. The upper limit of the temperature is determined by the phenomenon of 'overburning.' Overburned lime can be recognized by its yellow color and the extreme length of time it takes to slake, together with the fact that it is appreciably heavier than the white, properly burned lime. These properties are probably caused by the chemical combination of the lime with impurities (especially with silica and silicates) contained in it and the reduction in the percentage of pore space in the quick-lime. Lime may be overburned by being heated for too long a time as well as at too high a temperature. In general, it is better to underburn than to overburn the lime, for the unburned stone may be put back into the kiln while the overburned lime is useless. Moreover, the properties of overburning seem to assert themselves gradually, so that the best lime is obtained by using a minimum amount of heat.

Knibbs[21] attributes the character of the quicklime to several factors:

The properties of calcium oxide depend on the method of its formation, the temperature to which it has been exposed and the duration of the heating. . . . .

It also depends upon the physical properties of the carbonate. According to Kohlschutter and Feitknecht the primary particles of burnt lime are crystalline, preserving more or less the structure of the original substance, but on continued heating they aggregate to form secondary particles. . . . .

Knibbs[22] emphasizes the necessity of proper calcining temperature when he says:

The chemical properties of magnesium oxide depend, even more than those of calcium oxide, on the temperature to which it has been heated and the duration of the heating.

. . . . The oxide prepared by heating the native carbonate . . . . . to redness when mixed with a limited amount of water sets hard, but this property of hard setting is not shared by all oxide made at low temperature. Some low temperature oxide absorbs water, like calcium oxide, without setting. If the oxide is heated to a high temperature it loses the property of absorbing water, expecially if it contains a trace of iron oxide, when artificial periclase is formed . . . . . . Even the high temperature product, however, combines slowly with water.

21 - Knibbs, N. V. S., Page 48 of work cited.
22 - Knibbs, N. V. S., Page 74 of work cited.

Knibbs[23] warns against the overburning of magnesium limes since:

> Magnesian limes and magnesite are similarly subject to overheating, and the temperature at which they were dead-burned is much lower than that of pure lime . . . .

Since lime, in most chemical and all flotation uses, functions in an aqueous solution, the reactions in water are far more important to the ore dressing engineer than the fire reactions. The calcining should, therefore, be of such a nature as to produce a quicklime which is favorable to hydration.

The rate of hydration is determined by the physical and chemical properties of the quicklime which is being used. With subsequent hydration in mind, Knibbs[24] gives an explanation of the physical conditions of quicklime recently advanced, but the matter has never been fully investigated. The physical and chemical properties are closely related, for the physical modifications are due to chemical impurities in many cases. To quote Knibbs:

> In the operation of lime burning it frequently happens that lumps of lime are produced which when wetted hydrate only very slowly. Such lime is said to be over-burnt or dead burnt. It is usually easily distinquishable from good active lime by its appearance and its weight, white lime generally assuming a yellow tinge and becoming appreciably heavier when overburnt. The difference in the rate of hydration may be enormous. A good piece of lime on wetting may become hot and start to fall to a powder in a few seconds, whilst an over-burnt piece of lime may remain apparently unchanged for hours or even days. The term used for this slow-slaking lime implies the fact that its condition is due to overheating or to its remaining for too long a period at the high burning temperature, but an explanation of the change in the lime which so alters the rate of hydration has been suggested only comparatively recently, and the phenomena has never been fully investigated . . . .
>
> Since compact crystalline carbonate dissociates and the carbon dioxide escapes without any breaking up of the solid, which retains exactly the shape and size of the original carbonate, it is clear that the resulting oxide must be porous and of a very fine-grained structure . . . . In spite of its high melting point, lime softens at a comparatively low temperature, and it is clear that the exceedingly fine-grained structure which must be formed when the carbonate first parts with its carbon dioxide may rapidly change on continued exposure to heat. The change will obviously be in the direction of reducing the exposed surface of the oxide, that is to say, the size of the individual cells or pores in the lime structure will increase whilst the area of the enclosing walls will be reduced. This may continue for some time without any reduction in the total volume of the lump, but finally we should expect the cells to contract and the size of the whole lump to diminish, the percentage of pore space

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23 - Knibbs, N. V. S., Page 101 of work cited.

24 - Knibbs, N. V. S., Work cited, p. 99 et. seq.

being reduced. There is good reason to believe that these changes actually do take place under the influence of heat.

It must also be remembered that just as the solubility of a solid is affected by its surface conditions, so its chemical activity is similarly influenced, and an exceedingly fine-grained structure, which resembles a very fine powder, is more reactive than a coarse structure or a smooth surface.

In commercial lime we have also to take into account the impurities which are always present, and which reduce the activity of the oxide in two ways. In the first place, they are diluents. Silica, for example, combines with calcium oxide to form the compound $3\ CaO \cdot SiO_2$ and in addition to the three molecules which are thus rendered entirely inactive the calcium silicate acts as a diluent to the oxide. Moreover, at high temperatures either more basis compounds or solid solutions of the silicate with calcium oxide are formed, and the additional oxide so combined or associated with the silica is rendered entirely or partially inactive. Alumina and iron oxide act in the same way as silica. In the second place, these impurities increase the tendency towards the reduction of surface by reducing the temperature at which softening, and therefore contracting, takes place. The combination of these two effects accounts for the very considerable influence which the amount and type of the impurities exert on the relation between activity and burning temperature.

HYDRATION

Hydrated lime, the dry hydrate, is a produce of comparatively recent origin. It is lime in which the combination with water has already taken place. Holmes and Fink[25] say:

> Hydrated lime is now made by mechanical processes which vary somewhat, but all involve the thorough mixing of quicklime with the predetermined quantity of water required for complete hydration and maintenance of the most favorable conditions for hydration . . . . .

Until recently most of the lime used in the building industry and some of the lime consumed in the chemical industry was hydrated by hand. However, a good mechanical hydrator does the work better. Mechanical hydrators are of two types: Those that hydrate to a milk of lime and those in which the hydration is to a dry powder.

Lime sometimes burns when slaking. This is discussed by Porter[26], who says:

> The actual slaking of the lime takes place in an apparatus called the hydrator. Theoretically, the only function it has to perform is to mix the lime and water thoroughly and quickly, to prevent burning

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25 - Holmes, M. E., and Fink, G. J., Page 347 of work cited.
26 - Porter, J. M., Pages 68 and 69 of work cited.

the lime by local overheating. Just what happens when lime 'burns' during slaking is not well understood. If a lump of lime is given enough water to start hydration but not enough to complete the process, the unslaked portion will 'burn', and become practically useless. This phenomenon is probably due to some change in the physical condition of the lime caused by concentrating the heat. It is, therefore, absolutely necessary that sufficient water be added and that it shall be mixed with the lime very thoroughly and quickly . . . .

Whatever the system of hydration, the material discharged from the hydrator is much the same. Knibbs[27] states:

> Whatever the system of hydration, the material discharged from the hydrator is a mixture of fine hydrate and lumps of unhydrated lime, lumpy hydrate, impurities, unburnt stone, etc. Some manufacturers grind the lime very finely before hydration, but this practice is not to be recommended because it prevents the separation of good fine hydrate from impurities, etc., which should be the first step after the process of actual hydration . . . . .
>
> The fine hydrate is separated from the coarse material by means of either screens or air separators.

A booklet by the American Steel and Wire Co.[28] in discussing the slaking of lime recommends the use of hot water and not more than 4 pounds of water to 1 pound of lime. It drops the caution that the lime so slaked does not wholly loose its granular nature. It says:

> When quick lime is treated with water, heat is given off and the lime is reduced to paste or mud of lime. If core is present, lumps of it will be found unslaked. Not more than four pounds of water should be added to one of lime to slake it. If convenient, hot water gives better and quicker results than very cold water. In fact, very cold water should be used very carefully in slaking lime. If too much be used at one time, the lime will not slake properly. On the contrary, care must also be used in slaking lime with warm water, lest the lime get too hot and burn. When it gets too hot, more water should be added and the lime stirred with a mortarman's hoe. In case mechanical stirring devices are provided, the lime can be stirred mechanically and this is much better than using manual labor. Any soft, mushy lumps must be broken up to allow the water to get to the center of them. When the lime has ceased to bubble and absorb water, it has passed the danger stage and may be left till required. It will continue to undergo more perfect slaking if left to lie as a thick mud for a number of hours.
>
> When it has been slaked it has then become a hydrated lime, and is in much the same condition that a dry hydrated lime would be if mixed into a thick mud with warm water. There is this exception: a quick lime slaked as above does not wholly lose its granular character.

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27 - Knibbs, N. V. S., Pages 193 and 194 of work cited.

28 - Manual of American Steel and Wire Co.'s System of Water Purification; January, 1916, reprinted August, 1926; Chicago, Illinois, p. 48.

Lead or brass must not come in contact with lime. The same booklet[29] says:

Lime will quickly attack lead and dissolve it. Brass offers a better resistance, but is rather quickly destroyed, so it is not advisable to use any lead or brass in considering devices where they will come in contact with the lime solution. Wood offers an excellent resistance, and cast iron a still better one to the action of this reagent, and therefore these two materials afford us the best possibilities for construction of the stirring devices.

Knibbs[30] summarizes some of the chemistry of lime and hydrated lime:

Calcium oxide does not dissolve in water without first being converted into hydroxide. It therefore has no solubility in water . . . .

. . . . The solubility of the hydroxide has been determined many times, and there is considerable discrepancy between some of the figures given . . . .

The solubility of calcium hydroxide is increased by the presence of <u>chlorides</u> or <u>nitrates of the alkali metals and ammonium</u>, the increase being due probably to both double decomposition and the formation of complex salts (e. g. $2\ NH_4Cl \cdot Ca(OH)_2$ or $Ca\ (NH_3)_2Cl_2 \cdot 2\ H_2O$). It is also increased by the presence of <u>calcium chloride or nitrate</u>, and here the effect must be due to the formation of an oxy-salt . . . .

Calcium oxide, if pure, readily combines with water to form the hydroxide. Ordinary lime, made by calcining the carbonate, is porous and on moistening a lump of it the water is at once absorbed into the pores and rapidly penetrates to its heart. Combination then sets in accompanied by an increase in volume, so that the lump breaks up, sometimes with explosive violence, and eventually falls to powder

. . . .

Exposed to ordinary air calcium oxide combines with moisture and carbon dioxide, and if originally in lump form, it falls to powder (air slaked lime). Calcium oxide at ordinary temperatures and in the absence of water apparently does not combine with carbon dioxide, and in air slaking the first reaction is the formation of the hydroxide . . . . . Pure limes hydrate rapidly (complete in 10 to 50 days), and take up more water than is theoretically required for the complete conversion of the oxide to the hydroxide. Very little carbon dioxide is absorbed until hydration is complete, and the water content decreases as the hydroxide is converted into carbonate, but an excess of water is always retained, presumably absorbed on the carbonate or hydroxide. Magnesian limes absorb both water and carbon dioxide more regularly and more concurrently. Siliceous limes are variable . . . .

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29 - See footnote 28.
30 - Knibbs, N. S. V., Pages 39, 48, and 49 of work cited.

The properties of the hydrate vary with the method of making it. The metallurgist desires a hydrate that will remain in suspension for maximum reactivity. He can control this by regulating the amount of water used for hydration. If he buys the hydrate he should learn the method of preparation. Knibbs[31] says:

> The properties of calcium hydroxide vary considerably according to the method of its preparation and the nature of the lime from which it is made. Lime hydrated in an excess of water is generally more reactive than that made by adding just sufficient water to produce dry hydroxide . . . .
>
> . . . No satisfactory explanation of these facts has yet been given, and further investigation is required. It has been suggested that the hydrate formed by dry hydration is crystalline, or in some form of molecular aggregation that renders it less soluble. It is evident that there is some difference, but its exact nature remains to be elucidated. The hydroxide formed by hydrating in excess water is very slightly dispersed, and the solubility of the minute particles formed will be appreciably higher than that of the comparatively large particles of the dry hydrate, and the time taken by the former to saturate the solution will be less.
>
> Owing to its low solubility in water calcium hydroxide is generally employed in the form of 'milk of lime' when required for a reaction in an aqueous solution. Milk of lime is a syspension of the hydroxide in a saturated aqueous solution. It may be prepared in any strength, from an actual solution containing no lime in suspension to a thick slurry which will barely flow and containing about 50 per cent of hydroxide in suspension . . . . It is evident . . . . that lime hydrated to a dry powder is in a form that settles much more rapidly than that hydrated in excess of water . . . . . . Wet hydrate on drying partially loses its property of slow settling, and resembles that prepared in a dry way, but its properties depend upon the temperature at which it has been dried.
>
> Kohlschutter and Walther, and Kohlschutter and Feitknecht have shown that the rate of settling of the hydroxide is changed by hydrating the oxide in various solutions, and they measured the rate of sedimentation in a number of salt solutions of different concentrations. They conclude that there is a colloidal form of the hydroxide between the solid and the true solution . . . . . . Kosmann maintains that in milk of lime there exist various hydrates up to $Ca(OH)_2 \cdot 7H_2O$ or $H_7Ca(OH)_9$. It is certainly highly probable that there is some loose form of combination between the dispersed hydroxide and water, but it is doubtful whether any definite chemical compound is formed. The water molecules are probably merely absorbed. The particles of hydroxide prepared in a dry way are probably relatively large and in water the degree of dispersion is much less. Consequently

31 - Knibbs, N. S. V., Pages 50-53 of work cited.

the exposed surface is much less and the amount of water absorbed will be correspondingly lower. Such hydroxide tends to disperse on standing in contact with water . . . .

Magnesium limes may be expected to settle more quickly than calcium limes. This is mentioned by Holmes and Fink[32] who state:

The characteristics of milk of a lime suspension are of considerable importance in connection with the use of lime in many chemical and industrial processes. The rate of settling or permanancy of such a suspension is of particular and direct importance in such processes as the unhairing and plumping of hides, the preparation of glue and gelatin, the defecation of sugar juices, preparation of whitewashes and lime paints, the preparation and uses of sprays and insecticides, the treatment of water supplies, sewage disposal, treatment of textiles, and saponification and clarification of fats. In other processes involving the use of lime, such as causticizing, preparation of bleach and bleaching solutions and a number of processes in other industries, this property is of indirect importance in influencing the rate of reaction, rate of settling of sludges, etc. . . . . .

The data . . . . indicate that suspensions of magnesium limes as a class settle much more rapidly than do suspensions of calcium limes. This may probably be explained by the fact that in calcining a magnesium lime the temperature required to decompose the calcium carbonate is very much above that required for the magnesium carbonate. The resulting magnesia is therefore superheated and is not so thoroughly dispersed in the slaking of the lime. The suspensions of the material would consequently be expected in general to contain a large proportion of coarse material and to settle more freely than that obtained from high-calcium lime, which may be thoroughly disintegrated and dispersed.

From a consideration of the data . . . . . it can also be said that those limes, both calcium and magnesium, which as a class have a high available lime content, settle slowest. It would naturally be expected that the higher this value the greater the number of particles of the quicklime to undergo reaction during hydration . . . . .

A third generalization can be made - viz., that among the calcium limes the freshly slaked materials settle more slowly than the suspensions made from mechanically hydrated limes. As pointed out above, the conditions obtaining in the mechanical hydration are such as to favor high temperatures, which would tend to induce certain chemical reactions resulting perhaps in the formation of small quantities of 'oxyhydrate' which is described by Emley as 'a hard, gritty, yellowish material which shows a tendency to crystallize.' . . . . . .

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32 - Holmes, M. E., and Fink, G.J., Fundamental Properties of Commercial Limes; II - The Settling of Milk of Lime Suspensions: Chem. and Met. Eng., vol. 27, Dec. 20, 1922, pp. 1212-1216.

It will also be noted that the differences between the settling rates of hydrates are much less than the differences between the rates of settling of the freshly slaked material. Slaking with excess of water brings out the individual characteristics of the limes in this respect, while mechanical hydration tends to smooth out or reduce these differences. Extreme conditions in the rate of settling are therefore best obtainable with quicklime. On the other hand, the use of hydrated limes will result in greater uniformity. . . . .

The differences between the slaked magnesium limes and corresponding hydrates are much less than in the case of the calcium limes. The conditions obtaining in the mechanical hydration and storage, as compared with the slaking process, are probably more favorable for hydration of the magnesia, thereby minimizing the difference in the degree of dispersion and settling rate of the magnesium limes.

Knibbs[33] discusses the general influence of chemical composition. He shows that when more neutralization only is desired, magnesia is more effective than lime:

. . . .

The impurities commonly present in ordinary lime are silica, alumina, iron oxide, sulphate and magnesia. In many reactions magnesia is as effective as calcium oxide, and therefore it does not come under quite the same catagory as the other impurities . . . .

All the above-mentioned impurities except magnesia combine with calcium oxide, and silica and alumina which may not be combined with limestone generally combine completely with the lime during calcination. The composition of the compounds formed are not easy to determine, but they are generally assumed to be $3\ CaO \cdot SiO_2$, $2\ CaO \cdot Al_2O_3$, and $2\ CaO \cdot Fe_2O_3$. It is probable, however, that even more basic compounds than these are actually present in the lime, and this supposition receives support from the fact that the 'available lime' estimated in the usual manner . . . . . is not a measure of the value of a lime in all reactions, but in reactions such as those of water softening, sugar refining, causticizing, carbon dioxide absorption, and in general where there is no strong acid radical involved it fairly correctly reveals its true value. When a strong acid radical is present the compound of calcium oxide with silica, alumina, or iron oxide is generally attacked to some extent, and the percentage of lime actually reacting is greater than the 'available lime' obtained by analysis.

In many reactions magnesia is as effective as lime, and, in fact, weight for weight, it is more effective because of its lower equivalent weight. For example, where the lime is used merely to neutralize acid or where it is employed as a carrier, as in the sulphite liquor used in paper manufacture, the presence of magnesia is not detrimental. In the majority of its chemical uses, however, magnesia is either much less effective than lime (e. g. in causticizing soda) or is undesirable in the product made (e. g. in bleaching powder) or both.

33 - Knibbs, N. V. S., Page 219 of work cited.

The effect of incomplete calcination varies with different limes. A core or yolk of unburnt stone may remain intact after slaking, or it may break up partially or completely and contaminate the hydrate. It is important to use a lime in which the unburnt core, if there is any, remains intact for manufacturing of bleaching powder and other purposes in which low carbon dioxide content of the hydrate is desired.

Going further with the chemical properties of magnesium oxide, Knibbs[34] states:

The properties of the hydroxide have not been much studied. In general it behaves like calcium hydroxide but it is a weaker base and is much less soluble. It combines with carbon dioxide and acids in the same way as calcium hydroxide but with less avidity. A suspension of magnesia in water (milk of magnesia) absorbs sulphur dioxide and hydrogen sulphide . . . .

Although there seems to be some disagreement among authorities as to the proper method of hydration, they agree that the product is influenced by the method of adding water; and in general, they favor hydration with an excess of water over that required by theory. The advantages and disadvantages of magnesium in hydrated lime will receive further consideration.

LIME IN WATER TREATMENT

The purification of water by lime affords a study of chemical reactions that the ore-dressing engineer has to take into consideration when he uses lime in flotation. Lime will relieve temporary hardness by breaking up the bicarbonates, and soda ash will precipitate the ions of calcium and magnesium that remain as sulphates. This does not entirely purify the water because sodium sulphate is left in solution. The following excerpt from the National Lime Association bulletin[35] gives a more complete presentation of the chemistry and practice:

The elements of calcium and magnesium, in combination mainly as carbonates and sulphates, with small amounts of chlorides and nitrates, are responsible for the hardness of water. This hardness becomes objectionable when the salts are present in amounts sufficient to form a curd with soap, and a scale when the water is heated or evaporated.

Calcium carbonate and magnesium carbonate are practically insoluble in water, but since all neutral waters carry the gas-carbon dioxide, the weak carbonic acid which forms when CO_2 comes into contact with water, is capable of dissolving and holding in solution these carbonates of calcium and magnesium to produce the bicarbonates, or what is known as the 'temporary hardness' of water.

The water is softened by the removal of the carbonic acid either by heating, which is not a practical method, or by the addition of lime. When lime is added to waters containing dissolved limestone,

34 - Knibbs, N. V. S., Page 74 of work cited.
35 - See footnote 5.

it will absorb the carbon dioxide which holds it in solution, the lime itself being changed back to calcium carbonate, or limestone. The limestone so formed, as well as that originally found in the water, being deprived of their solvent, carbon dioxide, both become insoluble, settle out of the water and are removed as sludge.

Magnesium carbonate is slightly soluble in water and must therefore be converted to the more insoluble magnesium hydroxide before it can be properly precipitated. This reaction is brought about by adding additional lime, which produces magnesium hydroxide and calcium carbonate, both of which are insoluble and easily removed.

The sulphates of calcium and magnesium, known respectively as gypsum and Epsom salts, are both soluble in water and require the addition of soda ash, or sodium carbonate, for their removal. They produce what is called 'permanent hardness.' The soda ash, by effecting a double decomposition of the soluble sulphates, removes the calcium and magnesium by precipitation as carbonates, leaving the soluble sodium sulphate, which does not cause hardness, in the water. The small amounts of chlorides and nitrates which may be present are removed along with the sulphates.

It may be added that lime neutralizes acid, precipitates ferric sulphate, hastens the oxidation of ferrous sulphate, which in turn is precipitated. Organic matter inherent in the water is likely to be carried down by the precipitates.

Of interest to those who are confronted with the problem of liming mine waters containing iron salts is the observation that ferrous sulphate accelerates the purifying effect of lime. The main purpose of the booklet by the American Steel and Wire Co.,[36] already mentioned, is to promote ferrous sulphate as a purifier. It deals with water purification in a broad and thorough manner. The reading of the following excerpt from the section on "Alkalinities" may well be followed by a thorough study of the booklet:

. . . . . . It will be seen that alkalinity may be of three kinds, either caustic, monocarbonate or bicarbonate; that a caustic alkalinity will give a pink color to a phenolphthaleine test solution, and that the same color will be developed by a monocarbonate alkalinity, and that therefore the fact that a water turns pink when phenolphthaleine is added does not necessarily mean that caustics are present. Thus a water may have only a caustic alkalinity or the alkalinity may be wholly monocarbonate in type or it may be a mixture of the two. Or it may be wholly of a bicarbonate type or a mixture of bicarbonate and monocarbonate alkalinity. A caustic alkalinity and a bicarbonate alkalinity cannot, however, exist in the same water, and where free carbon dioxide is found neither caustic nor monocarbonate alkalinity will be present.

From the preceding it follows that six possible conditions of alkalinity might arise. The operator must be able to determine which one of these actually exists in any given water. He must also be able to

36 - See footnote 15, page 92 of work cited.

tell just how much of any type of alkalinity is present in any water as well as the total combined alkalinity of all types present.

These six possible conditions are as follows:

I. When caustic alkalinity only is present.

II. When monocarbonate alkalinity only is present.

III. When bicarbonate alkalinity only is present.

IV. When a mixture of caustic and monocarbonate alkalinity exists.

V. When a mixture of monocarbonate and bicarbonate alkalinity exists.

VI. When a mixture of bicarbonate alkalinity and free carbonic acid exists.

Some flotation reagents will permit the use of a variety of waters, for example, some alkaline reagents are known to work almost as well in acid as in alkaline water, but generally the chemical condition of the water has to be controlled between narrow limits. An example of the necessity of precise water treatment is the conditioning of water for soap as a flotation reagent. Oleic acid and the oleates are the most common reagents for the nonsulphides, and it is well known that they are made inert by hard water. The frequent references in the flotation literature to the degree of alkalinity indicate that a thorough knowledge of water treatment is required; lime is the outstanding reagent for this purpose.

LIME IN CYANIDING

Lime from Limestone

One of the principal determinants of the use of lime in ore dressing has been its abundance and cheapness. In some cases its purpose could be accomplished somewhat more efficiently by the use of a more expensive alkali, but the additional cost is not merited. In other cases the chemical properties of lime are more desirable than those of other alkalies.

It is not unlikely that the first extensive use of lime in ore dressing was in the cyanidation process, or in connection with it. At least we know that it was used extensively for two purposes, namely, neutralizing acid and settling slime. Its function in cyanidation is summarized by Clennell.[37]

> Lime is universally used as a neutralizing agent, and is frequently added in the battery to counteract the effect of soluble acid salts in the mill or mine water; it is also used as a coagulating agent in bringing about the settlement of slimes. It is added to the cyanide solution as a protection against the soluble and insoluble cyanicides in the material treated, and also against atmospheric carbonic acid . . . .

37 - Clennell, J. E., The Cyanide Handbook; New York, 1915, p. 537.

Since slime treatment is conducted at a higher pulp density than is often obtained from the milling operations, there must be a dewatering of the pulp between the mill proper and the cyanide plant. Such dewatering is accomplished by settling and decantation, but owing to the presence of amorphous clayey matter, it is often impossible to settle the pulp and obtain a clear overflow without adding some substance which will cause coagulation or flocculation of the clayey material. Hamilton[38] says about coagulation:

> . . . . The substance most commonly used for this purpose is lime both on account of its cheapness and also because of its usefulness as an alkaline protector for the cyanide. Many other substances have a similar effect though varying in degree, such as alum, calcium carbonate, sulphuric acid, ferrous and ferric sulphate, and calcium sulphate. Lime usually forms a coarser grained and more flocculent curd with a more brilliantly clear supernatant liquor than any other electrolyte, but the precipitated mass does not settle as densely as that produced by some of the others . . . .

Hofman[39] explains the precipitating action of lime by discussing the coagulating power of electrolytes:

> . . . . Coming to slimes proper, they may be said to consist of three classes of substances: Suspension, colloidal suspensions, and colloids (= pectoids of Cushman-Hubbard). A suspension is a finely divided substance stirred up in water, which will settle out gradually when left in repose; the addition of an electrolyte to the water has no effect upon the settling; the solid matter recovered by evaporating the water shows no change in its physical properties; the solid matter can be separated from the water by filtration. A colloid represents suspended particles which by carrying positive or negative electric charges repel one another continuously and thus counteract the clarifying force of gravity; the addition of an electrolyte neutralizing the electric charges causes the particles to coagulate, whereupon they settle out and carry down with them ordinary suspensions; the solid matter recovered by evaporation has lost its colloidal properties; the solid matter cannot be separated by filtration. A colloidal suspension represents a transition stage between the two extremes, and slimes belong to this category. As far as practical purposes are concerned, a slime may be defined as a colloidal suspension of finely crushed ore after 48 hours repose.
>
> The coagulating power of an electrolyte depends upon the valency of the kation. With the value of a univalent kation equal 1, that of a bivalent is 30 to 40, that of a trivalent 500 to 1,000. If 1 part FeO-salt settles 2,000 parts slimy water, 1 part Fe_2O_3-salt will settle 60,000 parts; 1 part CaO settles 1,500 to 1,800 parts. The figures "in the table below" give the relative weights required to produce the same coagulating effects.

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38 - Hamilton, E. M., Manual of Cyanidation: New York, 1920, p. 76.
39 - Hofman, H. O., General Metallurgy: New York, 1913, pp. 506-507.

Table . . . Efficiencies of various coagulating substances[1]

Substance	Relative Efficiency	Substance	Relative Efficiency
Aluminum sulphate	100	Alum (pot. chrom.)	858
Alum (potash)	143	Calcium chloride	1,095
Ferric iron	223	Calcium carbonate	1,215
Alum (ammonium)	252	Calcium sulphate	2,870
Alum (am. chrom. iron)	295	Magnesium sulphate	3,460
Lime	654	Sodium chloride	45,900
Magnesia	748	Sodium sulphate	61,700

1 - Julian-Smart, pp. 212, 220.

Fortunately, sodium sulphate, which is one of the most common by-products of water treatment, stands at the end of the list, showing that its coagulating power is not great.

Richard Meade[40] discusses hydrated lime as ideal material for use in cyanidation:

> There has been introduced in the lime trade, within the past few years, dry, slaked lime under the trade name 'hydrated lime'. This is simply calcium hydrate $Ca(OH)_2$, or in the case of certain limes, a mixture of calcium hydrate and magnesium hydrate, $Ca(OH)_2$ $Mg(OH)_2$. It is lime slaked with just enough water to convert it to hydrate without having an excess of moisture. When lime is slaked in the usual manner, water is added in considerable excess. This excess remains after the lime has slaked and forms it into a paste or putty. In the manufacture of hydrated lime, however, just sufficient water is added to slake the lime, and what small excess is employed is evaporated by the heat generated by the slaking process.
>
> There are a number of points of superiority possessed by hydrated lime over quicklime which make it peculiarly adapted to the use of cyanide plants. It has splendid keeping qualities and may be kept for several months without appreciable loss of value. There is absolutely no danger of fire from it, and it may be stored in wooden bins or even paper or cloth bags without any risk. It is extremely fine; 90% of it will usually pass a 200 mesh screen . . . . Hydrated lime is usually purer than lump lime, because in the process of manufacture impurities are screened out.
>
> On the other hand, of course, hydrated lime contains water, and hence a given amount will not contain as much calcium oxide as will the same amount of quicklime . . . . . .
>
> Generally speaking, hydrated lime is much purer than lump lime. This is particularly true of hydrates made by the air separator process.

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40 - Meade, Richard K., Hydrated Lime for Cyaniding: Eng. and Min. Jour., vol. 98, July 11, 1914, pp. 52-53.

In hydrates made by this method, the impurities are separated from the hydrate. These impurities consist of ashes, siliceous cores, unburned limestone and a small amount of unhydrated lime, the latter usually occurring from the over-burning and semifusing of lime . . . .

Meade's article shows that hydrated lime came to the cyanide industry before the year 1914 and that its properties were appreciated. Lime from dolomite will next be considered.

Lime From Dolomite

The Engineering and Mining Journal[41] gives the following discussion of various alkalies and the advantages and disadvantages of each. The discussion was prepared before lime came into use in flotation. It directs attention to magnesia and shows that an excess gives only a faint alkalinity.

> The problem of the neutralization of acid mine water has received only slight consideration, and a valuable agent, which promises to prove much more successful than lime (CaO) or carbonate of soda (Na_2CO_3), has been overlooked. This is magnesia (MgO). In using magnesite as a water purifier, the following facts may prove interesting: The combined weights of the substances (taking them all as normal) are: Sulphuric acid (H_2SO_4), 49; lime (CaO), 28; carbonate of soda (Na_2CO_3), 53, magnesia (MgO), 20. This shows that the theoretical amount of the alkalies required to neutralize 49 parts of sulphuric acid is: Lime, 28 parts; carbonate of soda, 53 parts; magnesia, 20 parts. It will be noted that in theory the magnesia is considerably more efficient, weight for weight, than either lime or soda. In practice this superiority is even more marked. Soda is superior to lime, owing to its greater solubility although theoretically lime is nearly twice as efficient as the soda.
>
> Sulphate of soda or Glauber's salt has an additional advantage over lime, owing to this solubility of its product; it does not precipitate and choke pipes, as the almost insoluble sulphate of lime (gypsum) does. The advantages of magnesia over lime are the same as those of soda, the sulphate of magnesia formed being very soluble. It has also an advantage over soda, magnesia being very insoluble in water, whereas carbonate of soda is readily soluble, thus causing loss in working. This advantage is apparent in practical working when a considerable excess of magnesia can be used to insure the complete neutralization without troubling to make careful tests of the acidity of the water, and this without any appreciable loss of magnesite. If the neutralization is carried out by causing the acid water to flow through a bed of calcined magnesite in lumps of suitable size, the work could be entrusted to an unskilled white man or even to a native, without danger of either incomplete neutralization or loss of magnesia,

41 - Engineering and Mining Journal, Neutralizing Acid Mine Waters: vol. 107, Mar. 15, 1919, p. 478. (Abstracted from So. African Min. Jour. and Eng. Rec., Dec. 14, 1918, p. 314).

it being merely necessary to replace the magnesite as it is dissolved away. When an excess of soda is used, this excess is lost, owing to its being dissolved unchanged and carried away. In addition to this, it will be noted from the combining weights that magnesia is more than 2.5 times as effective as soda, and as in practice 75 per cent carbonate of soda is found more efficient than lime, it follows that calcined magnesite, which can be easily obtained of a purity of from 90 to 95%, is more than three times as effective as lime.

Calcined magnesite has another advantage over crude carbonate of soda, inasmuch as it can easily be obtained practically free from organic impurities, whereas crude soda contains these to a considerable extent, and it is a well-known fact that even small quantities of organic matter in cyanide solutions interfere with the precipitation of gold in the extractor boxes. Calcined magnesite readily combines with sulphuric acid, even in dilute solution . . . .

Thus it is seen that MgO should not be tabooed entirely in ore dressing; it is the most active neutralizer of free acid.

The effects of magnesia in lime for the cyanidation process have been studied by Leaver and associates.[42] The results of their experiments indicated the following:

(a) In regular cyanide practice where cyanide solution is to be used repeatedly for extracting fresh batches of ore, the MgO content of calcined dolomite cannot be used advantageously to neutralize acidity.

(1) Where an excess of calcined dolomite is used to prevent the building up of magnesium salts in solution, the MgO content takes no part in the reaction.

(2) If just enough calcined dolomite is used to neutralize acidity, magnesium salts will build up and eventually form a saturated solution in which the cyanide loss would be high unless an unusual amount of 'bleeder' solution is run to waste.

(b) In the cyanide treatment of certain ores calcined dolomite may be substituted for lime with profit.

(1) Calcined dolomite is as effective as lime in preventing cyanide loss.

(2) Calcined dolomite nearly parallels lime in its precipitating and settling effect and causes no difficulty in filtration of the solution, or in the precipitation of silver by zinc.

(3) Calcined dolomite equals lime in the quantity of precious metals recovered.

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42 - Leaver, E. S., Davis, C. W., and Woolf, J. A., Calcined Dolomite as a Substitute for Lime in the Recovery of Gold and Silver by the Cyanide Process: Report of Investigation 2648, Bureau of Mines, 1924, p. 7.

(c) In treating certain ores by the cyanide process, the recovery of silver with the use of calcined dolomite as an alkaline agent is not equal to that obtained with the use of lime.

(d) The use of MgO content of calcined dolomite would therefore be confined to neutralizing acidity and aiding settlement where the ore is crushed in water for amalgamation prior to cyanide treatment and in cyanide practice where the solution could be discarded or 'bled' sufficiently to keep down excess of magnesium salts.

Calcined dolomite should not be purchased as a substitute for lime in the cyanide process without a thorough investigation as to its action on the ore to be treated and its effect on all operations connected with plant practice.

Because of the insolubility of magnesium hydrate and the solubility of magnesium sulphate the efficient use of dolomite is restricted. Except for special and well understood conditions, lime is safer.

LIME IN AMALGAMATION

Another use of lime which was developed through the cyanide process is the use of lime in amalgamation. When milling is done in cyanide solution all of the lime is usually added to the ore before it enters the grinding mill. The lime is pulverized and mixed with the ore and thus gives the protective alkalinity that is desired. Milling in pure water is seldom done except when amalgamation precedes cyanidation. Even in this case, lime is often added to the mill. When this method was adopted mill men objected on the basis that it would harden the amalgam. However, in some cases, due to the improved amalgamation of fine gold in the slime, a marked increase occurred in the amount of gold recovered on the plates.

Clennell[43] says of wet stamp mill operation, under the caption of "Application of Lime in Preparatory Treatment of Slimes:"

> In modern practice a certain amount of lime is usually added in the battery, sufficient to dissolve any grease from the crushing machinery which might fine its way onto the plates. This has the further effect of coagulating the slime and thus bringing it into more intimate contact with the amalgamated surface. Hence it has been observed that where lime is used in the battery, the extraction by amalgamation is higher, and the assay value of the slimy portion of the tailing is lower than when the same ore is crushed without the addition of lime . . . .

Since lubricating oil or grease is harmful in flotation as well as amalgamation, the use of lime as a "solvent" for grease, as suggested by Clennell, may be borne in mind. Although lime is not a true solvent for grease, it causes grease to form a less harmful emulsion.

43 - Clennell, J. E., Page 189 of work cited.

LIME IN FLOTATION

Lime Replaces Acid Circuit

Notable changes have been made in the flotation process in the past six years. The changes that have taken place are discussed by Thomas, Christmann, and Gifford:[44]

> The outstanding developments, prior to 1923, of the flotation process related particularly to improvement in the mechanics of the process. Reagents and their properties received minor attention. The circuits employed were usually of acid reaction, and flotation of the sulphides into a bulk concentrate was the object. This period of flotation may be termed the period of 'oil flotation.'
>
> The year 1923 marked the beginning of a new period in flotation, the period of 'chemical flotation.' Well characterized compounds such as copper sulphate, potassium xanthate, thiocarbanalid, and cyanide were substituted for the oil mixtures of the preceding period. A general substitution of the alkaline for the acid cirsuit occurred. The outstanding characteristics of these two periods in the evolution of the flotation process are:

Prior to 1923	1923
Oil flotation	Chemical flotation
Acid circuits	Alkaline circuits
Bulk concentration	Selective concentration.

Taggart[45] has the following to say about the abandonment of the acid circuit and the adoption of the alkaline circuit:

> Until within the last few years a majority of flotation operations were performed in acid pulps. At present, however, nearly all flotation is carried on in slightly alkaline pulps, as these are best for the chemical collecting agents. The change has effected large operating economies, since acid pulps, with copper ores particularly, were destructive of all iron with which they came into contact.

During 1923 coal tars and creosotes were the heavy oils most used in the flotation of copper ores, with pine oil as the frothing agent. The general practice was to use sulphuric acid and petroleum acid sludge to form acid circuits. The practice of acid circuits was, however, on the decline. Thomas Varley[46] writes:

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44 - Thomas, George C., Christmann, L.J., and Gifford, R. S., Hydrogen Ion Concentration - Its Control in the Flotation Process: Tech. Paper 11, American Cyanamid Co., New York , Jan., 1928, p. 1.

45 - Taggart, Arthur F., Handbook of Ore Dressing: New York, 1927, p. 845.

46 - Varley, Thomas, Consumption of Reagents Used in Flotation, 1923-1924: Report of Investigation 2709, Bureau of Mines, Oct., 1925, p. 6.

It might be said that during the latter part of the year 1924, flotation practice was undergoing a somewhat radical change, and many plants operated experimental units using alkaline circuits together with soluble frothing reagents, such as xanthates. As a result many have altered their practices materially, and at the present time great progress in metallurgical efficiency is being made.

Differential flotation practice, or the flotation of one mineral from another or others in the presence of gangue, is rapidly being perfected. This is done not alone by the use of oils and acids or alkaline reagents but by chemicals and modifying agents which are positive in their reactions on certain sulphide minerals while others are unaffected. Advantage is taken of these facts, and chemical combinations can be added which retard certain minerals very materially in their degree of floatability. It is desirable, of course, to retard undesirable minerals such as pyrite, and exclude them from lead and zinc minerals as well as from copper minerals, where the 'dropping' of pyrite can be effected without a sacrifice of a material amount of the desirable minerals.

Varley, again, for 1925[47] writes on the increased use of modifying reagents:

> In order to counteract the objectionable soluble materials present either in the ore or mill water, modifying reagents are used to 'condition the pulp' and by neutralization prevent interference of such soluble materials. Reagents of this type are soda ash, sodium silicate, sodium sulphite, crude soda, lime, and acids. In this connection lime is generally regarded as a 'chemical,' on account of its action on the minerals, and its reactions with acidic solutions or soluble salts in ore.
>
> The year 1925 saw the passing from acid to alkaline circuits. Chemical flotation agents and conditioning agents have proved far more effective when used in alkaline pulps. Lime, as compared with acid, is much cheaper, and can be handled and stored much more easily and safely than acid. Lime can be manufactured in almost any locality, and procured more readily than acid. Nearly all the big operators have their own lime plants now and several are manufacturing their own reagents.

The following table, compiled from data given in Reports of Investigations[48] of the United States Bureau of Mines shows the increased use of lime in flotation and also the general statistics of flotation operations. By comparing this table with Table 1, one can see that in 1928 flotation consumed 2.5 per cent of the total lime produced.

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47 - Varley, Thomas, Consumption of Reagents Used in Flotation, 1925; Report of Investigation 2777, Bureau of Mines, Oct., 1926, p. 2.

48 - U. S. Bureau of Mines Reports of Investigations as follows: No. 2203 for 1919; No. 2709 for 1923-24; No. 2777 for 1925; No. 2852 for 1926; No. 2931 for 1927; and No. 3004 for 1928.

Table 4. - Consumption of reagents used in flotation for the years 1919, and 1923 to 1928, incl.

Year	Tons treated by flotation	Ratio of concentrates	Pounds of lime	Pounds of reagents	Pounds of reagents per ton of feed
1919	26,545,564	8.55	1/1,570,200	113,510,234	4.2384
1923	37,811,044	13.04	2/2,422,491	161,377,789	4.2680
1924	45,105,101	15.28	2/2,053,600	178,699,681	3.9618
1925	45,490,331	16.301	2,166,819	81,666,967	1.7952
1926	50,889,254	15.177	162,240,359	201,711,795	3.963
1927	50,073,450	-	169,926,145	220,514,373	4.404
1928	59,064,385	-	208,249,403	264,033,473	4.470

1/ Kind not stated.
2/ For copper ores only.

An increase in the ratio of concentration followed the general adoption of the alkaline circuit. In 1919 the average ratio of concentration with the acid circuits was 8.55; for the following years the ratio of concentration in alkaline circuits was practically doubled. A slight decrease in later years, from 15.28 (on copper ores largely) may be due to the treatment of an increased tonnage in lead-zinc-iron ores.

With the almost universal adoption of the alkaline flotation circuits and in view of the abundance and geographic distribution of limestone from which lime is obtained, it is not to be wondered at that lime finds such extensive usage.

Weinig and Palmer[49] add the following on this subject:

> Lime is the most used alkaline reagent, because of its cheapness and availability. Caustic soda and caustic potash could be used to replace lime in many cases, but they are much more expensive . . . .

Concerning the wide usage of lime in flotation, Thomas[50] and his associates write:

> Since the abandonment of the strongly acid solutions in the flotation of copper, the lime circuit has been adopted almost universally . . . .

The full significance of the statement is realized when it is remembered that of the 50,073,450 tons of ore treated by flotation in 1927, 40,881,768 tons of copper ores, making only one concentrate, were treated. In 1928 more than 99 per cent of the copper ore milled was treated in alkaline circuits using lime.[51]

49 - Weinig, Arthur J., and Palmer, Irving A., The Trend of Flotation: Quar. Colorado Sch. of Mines, vol. 24, No. 4, Golden, Colorado, Oct., 1929, p. 22.
50 - Thomas, G. C., Christmann, L. J., and Gifford, R. S., page 4 of work cited.
51 - Miller, T. H., and Kidd, R. L., Flotation Reagents, 1928: Report of Investigations 3004, Bureau of Mines, June, 1930, p. 6.

Choice and Preparation of Lime for the Circuit

Since lime (CaO) plays its role in flotation in the pulp and hence in solution as $Ca(OH)_2$, it may be added to the circuit in three forms. S. E. Stein says:[52]

> Lime functions in the flotation process as calcium hydroxide, after being dissolved in the mill circuit water. It is added as (a) burnt lime--calcium oxide with other substances residual after burning of limestone, or (b) hydrated lime--dried calcium hydrate with other substances residual after the slaking of burnt lime, or (c) milk of lime--a suspension in water of particles of hydrated, or freshly slaked lime, making a mixture containing 25 per cent solids or less, depending on requirements. The water is of course saturated with respect to calcium hydroxide. Solubility of calcium oxide in distilled water, at 15° C., is 0.120 per 100.
>
> The effective portion of the added lime is that which goes into solution in the mill circuit water to react as calcium hydroxide, and its magnitude will therefore be governed primarily by the quality of the original limestone and the efficiency of the burning. Lime should, of course, be purchased on the basis of the available calcium oxide of standard analysis, as this figure has a reasonably dependable relative value; but it is worth noting that the figure thus given will usually be higher than realized in practice on account of the fact that solvent solutions in a mill circuit are not as good as those used by the chemist in his methods for this analysis.
>
> The choice between burnt lime, hydrated lime, and milk of lime will be governed by local factors. The problem to solve is that of getting the required quantity of calcium hydroxide into mill circuit solution in the cheapest way consistent with accurate control of the feed. Feeding of burnt lime or hydrated lime requires a greater refinement of mechanical control than that ordinarily used for feeding ores, if the proper flotation control is to be attained. A storage bin or hopper delivering onto a feeder of the conveyor belt type, the speed of which is controlled by a rachet mechanism, variable cone pulley, or variable speed motor has been used successfully where the feed from bin to belt could be maintained with respect to range of sizes. Milk of lime offers a surer feed control, and for that reason has been adopted at plants where the highest order of metallurgical results are demanded. For feeding milk of lime a storage tank equipped with means for maintaining the calcium hydrate particles in suspension is required, as well as variable-control fluid feeder.
>
> The adverse factor in burnt lime is the fire hazard wherever water shipment or long time storage is required, or air slaking in the latter case . . . . In most cases the payment of higher

52 - Stein, S. E., Lime in Flotation: Eng. and Min. Jour., vol. 125, Mar. 24, 1928, p. 487.

prices for hydrated lime will be found unwarranted, as a fairly good grade of burnt lime is usually available at the same source, from which a unit quantity of calcium hydroxide can be furnished to the mill circuit more cheaply than can be so furnished by the hydrated lime.

Robie[53] writes of one company's method of handling lime:

> Lime now being used extensively in the mill (about seven tons a day), apparatus for its local manufacture was installed some years ago. A kiln was constructed out of the old steel stack at the Shannon smelter, in Clifton. A good grade of limestone is available near by, and the entire operation of quarrying the stone and making the lime costs only about $6 per ton. About a quarter of a ton of fuel is required per ton of product. This contains about 90 per cent of total calcium oxide, and about 85 per cent of available. The burnt lime is crushed to 3/4-inch, stored in a steel bin which is drawn as required, and ground in the first Marathon mill that was ever made, secured from the old plant of the Detroit Copper Mining Company in the vicinity. Balls have been substituted for the rods originally used in this mill. The grinding is carried out in a closed milk-of-lime-circuit, the requisite supply for the concentrator being drawn off from the tank containing the circulating solution and introduced into the pulp at the following points: The primary screen undersize, the grinding mill feed, and in the concentrate and tailing launder. Arrangements were being made at the time of my visit to treat fresh water with lime directly as it comes from the kiln, eliminating the storage of unslaked lime.

Porter[54] gives the advantages of the use of hydrated lime as follows:

> The chief advantages to the consumer of hydrated lime are as follows: It can be handled more easily on account of its being in powder form; it will keep better than the lump lime for the reasons just noted; it does not require slaking, but must merely be soaked in water to prepare it for use. This saves time and labor and eliminates any danger of loss of lime due to unskilled slaking. Any unburned lime or overburned lime which has passed the sorter will not hydrate and can be screened out of the finished product. Hence hydrated lime should contain less refuse than the lump lime. On the other hand, hydrated lime contains 15 to 25 per cent of water, on which the consumer may pay freight.

Henderson[55] condemns dolomite and gives the following discussion on the use of lime to alkalize mill circuits:

> . . . . The question often arises as to whether calcined dolomite may not be substituted for strictly calcium lime for flotation purposes: There is but one answer, and that is the negative.

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53 - Robie, E. H., The Morenci Concentrator of the Phelps Dodge Corporation: Eng. and Min. Jour., vol. 126, Aug. 25, 1928, p. 293.
54 - Porter, J. M., Page 13 of work cited.
55 - Henderson, Clark T., The Proper Use of Lime to Alkalize Mill Circuits: Eng. and Min. Jour., vol. 120, Dec. 26, 1925, p. 1016.

The reason for this is that the lime, to be effective, must go into solution in the pulp, and magnesium hydroxide is scarcely soluble in water, so that if dolomitic lime is used, only the calcium hydroxide contained is useful for flotation purposes and much larger quantities must be employed.

Though lime is one of the commonest chemicals and seemingly one of the simplest, it has peculiar characteristics worthy of serious consideration in using it in flotation. As already stated, the calcium hydroxide must go into solution to be effective. A part of the lime which goes into solution is consumed in neutralizing the temporary hardness of the water used in making the pulp; more of the lime is consumed in neutralizing such acidity as there may be in the ore, and the balance creates the necessary alkaline condition in the circuit.

It is common practice in some plants to feed lime into the flotation circuit as dry quicklime, or calcium oxide. If fed thus, the conditions under which the quicklime will slake are quite uncertain, and the conditions under which the lime is slaked have a distinct bearing on its effectiveness in the flotation circuit. Other plants use hydrated lime, which is quicklime slaked with the theoretical amount of water required for hydration. This hydrated lime is either fed dry into the flotation circuit or is mixed with water to form milk of lime, which is then fed into the circuit. Still other plants make a practice of slaking quicklime with uncertain quantities of water, when making up a milk of lime of uniform strength for feeding into the flotation circuit.

A comprehensive series of experiments that were conducted in our company's laboratory gave the following results:

Where a certain amount of quicklime was slaked with the theoretical amount of water required for hydration; this hydrated lime then diluted with additional water to a standard volume; and this standard volume of milk of lime placed in a graduated cylinder, thoroughly agitated, and allowed to settle, the time required for the undissolved material to settle to 50 per cent of the total volume of the original milk of lime was 10 minutes. A corresponding sample of quicklime was next taken and slaked with 10 times the theoretical amount of water required for hydration. This was then diluted to standard volume, placed in a graduated cylinder, thoroughly agitated, and allowed to settle. The time required for the solids to settle to 50 per cent of the original volume was in this case 440 minutes.

It was obvious from the foregoing and other experiments that lime slaked with not less than 10 times the theoretical amount of water is much slower settling and consequently much easier to hold in suspension in the feed tanks, and has less tendency to settle out in the flotation circuit, than milk of lime made of hydrated lime, or quicklime slaked with small quantities of water.

Having thus proved that an excess of water gives a milk of lime having a lower settling rate, experiments were next undertaken to determine the relative chemical activity of these two types of milk of lime. Each sample of milk of lime was thoroughly mixed with the theoretical quantity of sodium carbonate that would react therewith to form calcium carbonate and sodium hydroxide. The relative time required for causticizing was taken as an index of the relative chemical activity of the two types of milk of lime. The milk of lime which settled in 10 minutes was found to require 330 minutes to effect complete causticizing of soda ash mixed therewith, whereas the milk of lime requiring 440 minutes to settle was found to react completely with soda ash in 5 minutes.

The foregoing experiments seem to prove beyond question that the proper type of milk of lime to use in flotation work is one made by slaking high calcium quicklime with not less than 10 times the theoretical amount of water required for hydration, because a milk of lime so made is not only slow settling but much more active in a chemical sense. The same conclusions would seem to apply to the use of lime to secure protective alkalinity in cyanidation.

Here, as in the section on "Hydration," the use of an excess of water in the preparation of the hydrate is recommended; and the evidence indicates that the hydrate made in this manner is much more finely divided. It is not, however, always practical for the operator to slake his own lime, in which case a good grade of hydrate may be used very effectively.

Place of Adding Lime

Lime may be added to a mill circuit in any of a number of places, or even in several places in the same circuit. Ralston and Yundt[56] point out the necessity of adding lime to the pulp far ahead of the flotation machines:

> . . . . It is probable that in the cases noted where lime improves flotation it functions in precipitating other less desirable soluble impurities, such as iron sulphates. At the Miami mill it was found that when lime was added to the pulp before entering the ball mills an increased recovery was obtained, whereas the addition of lime at the head of the flotation cells resulted disastrously. Supposedly the lime had to be allowed time enough to react with all the soluble or semi-soluble impurities in the pulp before going to the flotation machine, so that there would be no raw caustic lime to cause trouble.

Robie[57] states that at Morenci ground milk of lime is added to the primary screen undersize, the grinding mill feed, in the concentrates and tailings launder, and to the fresh water as it comes from the mine.

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56 - Ralston, O. C., and Yundt, L. D., Chemicals Used in Flotation; Min. Sci. Press, vol. 115, Oct. 13, 1917, p. 547.

57 - Robie, E. H., Page 293 of work cited.

Tye[58] reports adding milk of lime and a small stream of tailings to the mill water supply, so that the impurities are precipitated as gelatinous salts which are buried by the slime and sand of the tailings. To overcome the effect of soluble salts in the ore and obtain the desired alkalinity for flotation, more lime is added at the rod mill.

MacDonald[59] cites an instance where "dry lime is fed mechanically onto the ore conveyors . . . to the primary mills."

McDermid[60] in a recent article discusses the new practice at the Miami Copper concentrator where they

> . . . . . emulsify the lime first and then pump it to the mill pond, where the insoluble material would have a chance to settle out.

Regarding the removal of the precipitated impurities, he writes:

> . . . . . After some experimenting a dragline scraper was built which successfully removes the sludge from the bottom of the pond.

Commenting on the practice in the Tri-State District, Anderson writes:

> In the Tri-State District lime (hydrated) is added for its beneficial effect on flotation results at anyone of a number of places in the mill. Often it is added to the ball mill feed, sometimes to the waters going to the Dorr thickener, sometimes to the pulp passing to the head flotation circuit; occasionally it is added to the settling pond or to the mill pump, well, or lake. The general practice seems to be to add it at some place in the mill where it will have time to act and exert its effect before the pulp reaches the flotation machines. Several methods of adding lime are in use; adding by hand is practiced often, and adding by means of several homemade devices is quite prevalent. Most of these feeders are some form of hopper-like boxes in the bottom of which is a worm screw or some simple agitating device for causing the lime to flow. Recently one mill installed a continuous belt feeder which has proved very satisfactory. Very often lime is not required continuously, hence it is added one week and not the next. Lime is added usually as the hydrate.

Lime is not used much in the lead belt of southeast Missouri. If thickening capacity is limited, a little lime is used to aid settling, but caution is required lest the lime retard the flotation of lead. In the differential lead-zinc plants of the district, hydrated lime is used at the head of the zinc circuit to depress the iron. The price paid for hydrated lime is \$9.40 per ton.

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58 - Tye, A. T., Differential Flotation of Copper at Cananea: Eng. and Min. Jour., vol. 121, Apr. 10, 1926, pp. 597-602.

59 - MacDonald, Wm. T., Selective Flotation at Nacozari: Eng. and Min. Jour., vol. 118, Sept. 20, 1924, pp. 445-454.

60 - McDermid, A. J., Liming Mill Water: Eng. and Min. Jour., vol. 130, Oct. 9, 1930, p. 341.

Lime as a Purifier

The waters encountered in milling and flotation operations are often highly contaminated with soluble metallic salts and must receive special treatment to make them fit for flotation. An unusually complex water problem was encountered at Cananea (Mexico). The difficulties encountered and the method of overcoming them are given in the article by Tye.[61]

Differential flotation of a copper ore containing large amounts of pyrite offers no unusual difficulties when the ore is unoxidized and when fresh water is available. At Cananea, however, several elements unit to interfere with the process, and therefore the following account will deal principally with these unusual features and give in detail the methods used in overcoming them . . . .

The following analyses show the effect of lime on the treatment of dam water, a sample of which was made alkaline by the addition of lime, using phenophthalein as indicator:

Dam Water Analyses in Parts Per Million

	Ferrous	Ferric	CaO	SO_3	Cu	Alkalinity (CaO)
Dam water	400	200	470	2,270	28	(Neutral to (phenol- (phthal- (ein
Same, treated and filtered	Tr.	Tr.	1,420	2,106	17	

The precipitate gave the following analysis in per cent: Fe, 20; S, 5.8; Mn, 11.0; Cu, 16.5; CaO, 3.4; SiO_2, 3.2; and Al_2O_3, 3.0.

One can appreciate better the real nature of this water after having seen it made alkaline with lime. At first a light yellowish flocculent precipitate develops as a small amount of lime is stirred in. Then, as the final amount of the required lime is added, the whole mass of water takes on a deep bluish-black color and the consistency of the water approaches that of thin soup. The manganese salt is the last to settle out, as a pink precipitate. Finally the floccules settle, leaving a clear liquid of the analysis shown above . . . .

By the last of April everything was ready to try the process in the concentrator. For the first month and a half the results were quite poor, owing to the impure water. The tailing dam from which the water was drawn was filled with the impure water and the supply was augmented by the mine water, which was equally bad. The water was neutralized with lime before being used, but it still contained the precipitated iron salts, and no acceptable results could be obtained. This trouble was partly overcome by utilizing all the extra Dorr tanks, of which there were several, to settle out these precipitated salts.

61 - Tye, A. T., Work cited.

The water for the dam was lime-treated, mixed with tailing, sent to the tanks, and the clear overflow of the latter repumped for mill water. This cut the concentrator tonnage in half, as sufficient water could not be supplied. Therefore, means were provided to add excess lime to the tailing going to the dam to purify all the water in storage, as it was realized that in this way only could the process be applied successfully. This consumed considerable time, and it was June 16 before all the storage water was purified. Since then it has been necessary to use sufficient lime to secure the desired alkalinity for flotation and also to purify the makeup water which comes from the mines. This latter is treated before it enters the tailing pond.

In the meantime, the worst mine water was being segregated and discharged outside the drainage area. The change made in the composition of the great bulk of water in the tailing pond is shown in the following analyses:

Dam Water, Parts per Million

Date	Ferrous	Ferric	Mn	CaO	Cu.	SO_3	Alkalinity	Acidity
Feb. 12, 1924	600	120	50	420	42	2,559	Nil	40
June 16, 1924	Nil	Nil	Nil	1,260	Nil	1,749	50 M.O 39 P.P.	--

. . . . When passing the concentrator, however, milk of lime is run into the arroyo water until it shows distinct alkalinity. Crushed lime is fed into a small ball mill, and a minimum amount of water added while grinding so that considerable heat is generated in the mill, the lime does not get 'drowned,' and the majority goes into solution. This is run into the middle of the stream of water to be treated, and at the same time a small stream of tailing is also discharged with the lime, so that the gelatinous salts are precipitated and the slime and sand of the tailings bury them. At a point below where the lime is added to the arroyo water, a sample is taken every hour and titrated to be certain that sufficient lime has been added.

To overcome the effect of soluble salts in the ore (and to secure the desired alkalinity for flotation), lime is added to the rod mills, the discharge from these being tested every hour. Flotation feed is titrated for alkalinity hourly . . . . . The lime used is burned locally by the company and costs $6.50 per ton . . . .

Best results are with an alkalinity in the flotation feed equal to about 0.55 lb. of CaO per ton of water . . . .

Flotation reagents used are potassium xanthate, steam-distilled pine oil, and lime, the average pounds per ton being 0.09 xanthate,

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62 - Gaudin, A. M., Control of Flotation; Eng. and Min. Jour., vol. 125, Mar. 10, 1928, p. 418.

0.23 pine oil, and 9 to 11 lime. The lime includes that used in treating the mine water and varies considerably from month to month, depending on the purity of the water and character of the ore. In extreme cases it has averaged 12.5 lb. for a month . . .

. . . .

Mill waters vary widely in composition from time to time, and are one of the most pertinent factors affecting the continuity of flotation results. Concerning this subject, Gaudin[62] writes:

> . . . . . Variations in mill water are due to somewhat different causes, according as the mill is using fresh water or returned-circuit water. Where fresh water is used, the change in mineral and organic content is one of the big upsetting factors--particularly true in places where climatic conditions affect the amount of water available, where periods of oxidation of the ground alternate with periods of heavy precipitation. When mine water is used, variations are also likely to occur as new sections of the mine are opened up.
>
> . . . .
>
> If the water is returned to the mill after use--where circuit water is the dominant ingredient--the water is usually more regular in its mineral content than if fresh water only is used. On the other hand excess reagents can build up rapidly under such conditions, so that accurate control of the amount of reagent is still more essential . . . .

McKay[63] sums up the soluble metallic salts generally present and their removal from contaminated water in the following manner:

> The most common soluble metal salts are generally the salts of the minerals, including sulphates of zinc, copper, and iron; and salts derived from the gangue, such as sulphates of calcium, aluminum, and magnesium. However, in flotation circuits, many salts often thought of as insoluble enter solution in extreme dilution. The importance of even the weakest solution of such salts will be appreciated by remembering that many of our reagents--notably the promoters--are extremely effective even when present to the thousandth of a pound per ton of ore, which is equal to one part in ten million, or 10^{-7} (at 20 per cent solids). At this dilution many of the so-called insoluble salts are in solution in sufficient concentration to influence flotation conditions, so that a list of the soluble salts and also the reagents used to combat them would appear like an abbreviated chemical dictionary. The common polar reagents for destroying the toxic constituents of the ore are lime, soda ash, and sodium sulphite, sulphide, and silicate.

The simplest case of the removal of soluble salts from solution by the use of lime is in the purification of water for industrial purposes. (See section "Lime in Water Treatment.")

63 - McKay, Nevin Hall, Soluble Salts as Flotation Reagents: Eng. and Min. Jour., vol. 128, Dec. 14, 1929, pp. 920-921.

Knibbs[64] writes of the softening of water for domestic purposes and shows that lime precipitates organic matter:

> Water for domestic purposes may also be subjected to a softening process, but it is still more important to free it from organic matter and from harmful organisms. Precipitation of the hydroxide of iron and aluminum tends to carry down the organic matter, and lime is used in this connection to neutralize acidity and effect precipitation. Lime itself may be sufficient to precipitate the bulk of the organic matter. . . . . .

Hahn[65] adds the following about ferrous, ferric, and aluminum sulphates and their bearing upon copper recoveries, and their removal:

> . . . . The influence of the salts of iron and aluminum, and methods for eliminating these effects in the flotation of copper ore have been the subject of an investigation to be summarized here . . . .
>
> The following deductions are drawn from the results obtained:
>
> 1. Ferrous, ferric, and aluminum sulphates seriously affect copper recoveries, even when present in relatively small amounts.
>
> 2. Burnt lime eliminates, to a large extent, the deleterious effects of ferric and aluminum sulphates, but offsets to only a small extent the effect of ferrous sulphate.
>
> 3. By adding an oxidizing agent, such as oxygen, chlorine, or chloride of lime, to the ore pulp in the presence of burnt lime, the effect of ferrous sulphate can be almost completely eliminated.

Ralston and Yundt[66] say the following about the use of calcium sulphate or the effect of it in the flotation circuit:

> Calcium sulphate is a compound which has been added to ores containing colloidal gangue, although its success has been somewhat erratic. It was once used in one of the Broken Hill mills and its effect was that of an electrolyte. It is sparingly soluble, so that there can never be a high concentration of its ions in solution and it is hence more or less equivalent to the tartrates in providing ions rather slowly, so that, supposedly, the flocculation of gangue-slime will not entrain particles of the desired mineral.

Instead of emphasizing the use of lime, McKay[67] calls attention to ferrous sulphate as a flotation reagent:

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64 - Knibbs, N. V. S., Page 272 of work cited.
65 - Hahn, A. W., Obviating the Harmful Effects of Soluble Salts in Flotation: Eng. and Min. Jour., vol. 123, Mar. 12, 1927, p. 449.
66 - Ralston, O. C., and Yundt, L. D., Page 549 of work cited.
67 - McKay, N. H., Page 921 of work cited.

Again, without intending to be bound by any one theory, I believe that in copper-iron separation, ferrous sulphate, generally native in the ore, plays a part analogous or parallel to the action of zinc sulphate in lead-zinc flotation. As to the end reaction of excess ferrous sulphate with xanthate, mill experiment has not yet given data to show whether iron xanthate could be used in copper-iron flotation as zinc xanthate has been used in lead-zinc separations. Apparently sulphated pyrite, which occurs in nearly all copper-iron or lead-zinc-iron flotation, does play a part in the formation of ferrous or ferric sulphates, or cyanides of ferrous or ferric iron, which act in iron depression. The practice here, therefore, has naturally developed of using commercial copperas or iron sulphate with the sodium sulphite or cyanide as a pyrite depressant, with results that have been highly gratifying, yielding again a higher-grade concentrate, higher recovery, and smoother metallurgical performance.

Another use of lime in flotation which is closely related to precipitating soluble salts is its use in producing coagulation and settling of colloidal particles of slime and the "sweeping down" of suspensions of organic colloids. This action is a physico-chemical one. Rickard and Ralston[68] discuss the subject as follows:

> A word must be devoted to the use of lime as a flocculating agent . . . . . At first sight it would appear improbable that lime could have a purely physical flocculating action, lime being an alkali and the characteristic effect of small concentrations of alkali appearing to be a deflocculation rather than the reverse. There is, nevertheless, a certain flocculating action exhibited by lime, even under laboratory conditions, and it appears that the hydroxides of calcium, and probably magnesium do not behave exactly like potassium and sodium hydroxides. It appears probable, also, that certain purely chemical factors enter as well and that the efficacy of lime as a clarifier is due in part to chemical reactions of the same sort as those that control, for instance, the action of aluminum salts in clarifying water. These clarifying actions apparently depend upon the formation by chemical reaction of some flocculent precipitate that entangles and sweeps down suspended particles. Probably the occasional cases of clarification by organic colloids belong to the same class.

Ralston and Yundt[69] add:

> Lime is not such a desirable addition-agent as the alkaline sodium compounds; and while it may cause desirable effects when added in small amounts, an excess is often harmful. Lime and most other calcium compounds are usually flocculators of gangue-slime, rather than deflocculators as the alkaline sodium compounds are. This may explain the difference between the two, although it has

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68 - Rickard, T. A., and Ralston, O. C., Flotation: San Francisco, 1911, p. 320.
69 - Ralston, O. C., and Yundt, L. D., Page 547 of work cited.

been hinted often that we know too little about the degree of flocculation of the ore-pulp during flotation . . .

The settling effect of lime is mentioned by Weinig and Palmer:[70]

There are many factors that influence the settling rates of finely ground ores. For instance, an ore that has been air dried settles more rapidly than one coming directly from the mine. Argillaceous ores have poor settling properties. Lime and xanthate promote settling, while soda ash and sodium sulphite retard it.

Thus lime plays a multiple role as a purifier. It may be used to remove most of the impurities in water. Through chemical action it brings about the precipitation of the salts that cause temporary hardness and also precipitates many of the metallic compounds. Through physical action it facilitates the settling of suspensions and aids in the removal of organic matter.

LIME FOR ALKALINITY

The advantages of alkaline flotation circuits allow the use of an excess of lime in the purification of mill water. That which remains is active in producing alkalinity and becomes an important flotation reagent. Henderson[71] mentions the use of excess lime in solution as follows:

. . . . A part of the lime which goes into solution is consumed in neutralizing the temporary hardness of the water used in making the pulp; more of the lime is consumed in neutralizing such acidity as there may be in the ore, and the balance creates the necessary alkaline condition in the circuit.

Earlier in the same article Henderson says:

Xanthate will give good results in acid circuit, but it seems more satisfactory in alkaline circuit, and the advantages of flotation in alkaline rather than acid circuit are usually such as to make the alkaline circuit preferable, even though the metallurgical results in both are the same.

Alkalinity can be best secured with the aid of lime, as is done in cyaniding, the quantity required varying in practically every instance . . . .

The fact that the quantity required varies, makes alkalinity control important.

It is quite evident that the calcium ion and the hydroxyl ion both play a part in flotation, though the action of both is subject to uncertainty. Stein says:[72]

70 - Weinig, A. J., and Palmer, I. A., Page 43 of work cited.
71 - Henderson, C. T., Page 1016 of work cited.
72 - Stein, S. E., Page 489 of work cited.

. . . . It is necessary to know a great deal more about the action and effects of the calcium ion; and this may prove a difficult problem. With respect to the hydroxyl ion it is probable that experimentation with various reagents used in flotation under careful pH control will result ultimately in important advances in our knowledge.

Stein, earlier in the same article, cites two instances of the control of alkalinity.

One plant treating a fairly clean sulphide ore, as indicated by its low burnt-lime consumption of 1.5 lb. per ton of ore, used a standard of alkalinity for the flotation heading water equivalent to 0.20 lb. calcium oxide per ton ore (at 20 per cent solids). Markedly bad effects were noted when this quantity fell below 0.10 lb. CaO, but the facts are uncertain as to the intermediate range. Samples were taken at half-hour intervals.

Another plant, having a lime consumption of approximately 5 lb. per ton of ore, used a standard equivalent to 0.10 lb. CaO per ton of ore, at 20 per cent solids. Titrations were made at 15-min. intervals. The bad effects were visible to the operator when the alkalinity declined to 0.07 lb. CaO, and the appearance of a couple of '0.05's', and say, one '0.00' on the titration record sheet was an infallible sign that the corresponding assay sheet would indicate inferior metallurgical results. The effects of 'overliming' certainly were not visible to the operator until an alkalinity of 0.15 to 0.17 lb. CaO was reached, and inferior metallurgy resulted with certainty only when a considerably higher figure was reached . . . .

Among the various ores treated at one company's plant there would occasionally be received one which laboratory tests indicated was amenable to flotation treatment, with approximately equal results, by the use of either lime, or sulphuric acid, or in a 'neutral' circuit. But the rule was that either alkaline or acid circuit, each with its special accompanying reagents, was required to produce the best obtainable metallurgy. The 'neutral' circuit would show material losses in the tailing. There is, of course, the possibility of existence of a suitable reagent for the 'neutral' circuit, but this was not discovered.

The control of alkalinity presupposes an accurate method of quantitative determination. Phenolphthalein, the indicator frequently used in cyanidation and flotation, is not sensitive toward low alkalinities. Its change in color occurs only when the solution is well on the alkaline side (pH 8.5), and the fact that it is colorless in pulps more acid makes it useless near the neutral point. It is interesting at this point to give the discussion of Thomas, Christmann, and Gifford[73] on the difference between hydrogen ion concentration and acidity as expressed in normality:

73 - Thomas, Christmann, and Gifford, Pages 2 and 3 of work cited.

It is necessary to distinguish between the hydrogen ion concentration of a solution and its acidity as expressed in normality. A liter of normal acetic acid will, of course, neutralize the same amount of alkali as a liter of normal hydrochloric acid, but the hydrogen ion concentration in the latter solution is much greater than in the former. This is the reason for many of the differences in the properties of acetic and hydrochloric acids and is what is usually meant by the statement that hydrochloric acid is 'stronger' than acetic acid. The hydrogen ion concentration is, therefore, a measure of what might be termed the 'active acidity' of a solution rather than of the total acid concentration.

. . . It is obvious that the pH of a solution cannot be determined by titration, for this would show the total acid or alkali available for neutralization, which, as we have seen, is not a measure of the hydrogen ion concentration. Furthermore, the introduction of acid or alkali into a solution would immediately change the pH value and thereby destroy the thing to be determined.

Regarding the function of lime alkalinity, MacDonald[74] says the following:

. . . . . . Lime assumes the important function of protecting xanthate by impeding its precipitation by soluble iron, though happily permitting the corresponding reaction with soluble copper. Sodium sulphide, or even an excess of sodium xanthate itself, might well serve some functions of lime, except as the wastage of these more expensive reagents by soluble iron might be intolerable. The discriminating precipitation of the latter is desirable. In a lime circuit, the relatively low solubility of calcium xanthate may be a disadvantage.

Thomas and his associates[75] add the following remarks concerning the use of aerofloat in a pulp containing high lime alkalinity:

. . . . . When used in solutions of high lime alkalinity, Aerofloat has occasionally given erratic results and this fact has led us to investigate the effect of alkalinity upon flotation, with particular reference to flotation in solutions whose reactions are very near the neutral point. . . . . .

Stein[76] adds:

. . . . One large company that found it profitable to adopt pH control as a guide to flotation operations discovered that the correct value when using one reagent was not the best value when using another. Thus when it discontinued the use of the lime-xanthate circuit in which the correct pH value was approximately 10, it was found that the lime-aerofloat circuit, subsequently introduced, operated to best advantage when the pH was 8.3.

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74 - MacDonald, Wm. T., Selective Flotation: Eng. and Min. Jour., vol. 126, Oct. 27, 1928, p. 681.

75 - Thomas, Christmann, and Gifford, Page 1 of work cited.

76 - Stein, S. E., Page 489 of work cited.

Thomas and his co-authors[77] think that lime is not needed when cyanide is used as a depressant:

> It may be mentioned in passing that the depressor effect of cyanide is almost always brought out to the best advantage by working in solutions much on the acid side of phenolphthalein. The use of an alkali to 'protect' the cyanide, following the precedent of ordinary cyanidation, is not only unnecessary but is actually detrimental to the effect of cyanide as an iron inhibitor.

Stein[78] writes of the effect of over and underliming:

> With regard to the comparative deleterious effects of 'underliming' and 'overliming', a more thorough investigation as well as the collection of available data from diverse sources is needed. For what it may be worth, I am herewith setting forth my present impression on this subject. It is probable that the theoretically best degree of alkalinity varies for each mine, and consequently also varies with ores originating at different points in the same mine. The necessity of constant experiment is indicated.
>
> In starting operation with lime it would be wise to set the standard point of alkalinity sufficiently far in advance of neutrality--consistent with the avoidance of excessive overliming--to reduce the recurrence of complete loss of alkalinity to a minimum. 'Underliming', especially when it results in complete loss of alkalinity, is markedly harmful to metallurgical results. The froth livens up, becoming decreasingly selective in appearance as the alkalinity diminishes. If the condition is not remedied by speedy lime additions, the froth will 'run away' in spite of attempts at correction by reducing the quantity of the frothing agent. This naturally affects both the concentrate and the tailing.
>
> 'Overliming' has the reverse tendency unless carried excessively far, when a condition in outward appearance something similar to 'underliming' will take place. The first 'overliming' slows the froth down slightly, and thus has a tendency to clean up the concentrate with slight loss in the tailing. This moderate 'overliming', however, is not usually harmful, as it is easily corrected. The corresponding degree of 'underliming' usually would show a definite loss in tailing for the shift. However, at one plant which treats a variable ore, I have noted on three separate occasions the loss of a considerable quantity of the standard alkalinity, without a manifestation of the bad 'underliming' effects as described.

Thomas, Christmann, and Gifford[79] write of the amount of lime generally used and the effects:

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77 - Thomas, Christmann, and Gifford, Page 6 of work cited.
78 - Stein, S. E., Page 488 of work cired.
79 - Thomas, Christmann, and Gifford, Page 4 of work cited.

. . . . The amount of lime employed is generally in excess of that required to neutralize the true acidity of the ore and the resulting circuit is usually strongly alkaline toward phenolphthalein. These high alkalinities require the use of large amounts of lime, so that the cost of lime is frequently larger than that of the other reagents combined. As a reason for these high lime additions, the depressant action of lime upon flotation of pyrite is frequently cited. While it is undeniable that lime does inhibit the flotation of pyrite, the depressant action is not specific to pyrite and the advantage gained in this direction may easily be lost in others. Thus high lime alkalinity has been found detrimental to the flotation of chalcocite; it has also been found to cause irregularities in the behavior of the flotation reagents, notably of frothers.

The neutralizing and depressing effect which lime exhibits is probably the chief reason for its use as a flotation reagent.

LIME AS A DEPRESSOR

(A). Comments on Depression

There are two theories to account for the depressing action of lime: (1) The slime theory, and (2) the theory involving preferential deposition of a compound formed by the added agent and some constituent of the ore pulp other than the mineral upon which the salt is deposited (calcium carbonate theory of Gates and Jacobsen).[80]

The effect of lime in the slime theory is discussed by Taggart, Taylor, and Ince.[81]

The presence of certain soluble inorganic compounds causes amazing differences in the behavior of the gangue slimes. Thus lime in the proportion of 500 mg. per l. (equivalent to about 4.0 lbs. per ton of ore in the usual pulp) substantially prevents deposition of Anaconda slime on galena . . . . . . The same is true with chalcopyrite although this latter mineral, like galena, is heavily coated in the absence of lime . . . . . . On the other hand, when lime is added to a quartz-slime pulp, deposition on the sulphides is markedly enhanced . . . .

While lime is the reagent whose effects have been pictured, additional experiments have proved that it may be taken as typical of the other inorganic reagents that have been used in ordinary collective flotation. Sodium carbonate, sodium bicarbonate, sodium hydroxide, potassium tartrate, sulphuric acid, alum, barium chloride, etc.,

80 - Gates, J. F., and Jacobsen, L. K., Some Flotation Fundamentals and their Potential Application: Bull. of Univ. of Utah, vol. 16, No. 4, Aug., 1925, p. 38.

81 - Taggart, A. F., Taylor, T. C., and Ince, C. R., Experiments with Flotation Reagents: Am. Inst. Min. and Met. Eng., Tech. Pub. 204, Class B, Milling and Concentration, No. 17, New York, Mar. 1929, pp. 45-9

have been found to have similar effects, differing with particular sulphide and gangue present.

. . . . Addition to the Anaconda-slime pulp of cyanide and zinc sulphate (the bicarbonate was independently proved to be not effective alone) prevented any considerable slime deposition on galena . . . . . . without materially changing that on the sphalerite . . . . . . Pyrite also has a fairly heavy coating . . . . . in the presence of these reagents. Under relatively weak flotation conditions galena might be expected to float away from blende and pyrite in this pulp. Subsequent addition of more oil together with copper sulphate caused little change in the blende coating . . . . although visual inspection indicates a slight reduction, but it does cause a marked increase in the coatings on galena . . . . . and pyrite . . . . Hence flotation conditions can be intensified sufficiently to float the blende without effecting any considerable raising of galena and pyrite.

The film formed by lime appears on the surface of the minerals somewhat like that obtained from a sulphide film formation, but the method of formation is of course different. Gates and Jacobsen[82] explain this as follows:

The effect of addition reagents such as lime and copper sulphate on the floative properties of galena, sphalerite, and pyrite, suggested the possibility of surface coatings or precipitates being deposited on the surfaces of the minerals. The reflecting microscope was freely used to determine the extent to which the surfaces of the different minerals had been affected by the treatment received, and proved to be of great aid in substantiating the existence of surface coatings. The coating produced by lime has a depressive effect on the three sulphides, and that produced by copper sulphate an accelerative effect on the flotation of sphalerite and pyrite. Photo-micrographs actually revealed superficial coatings deposited on the mineral surfaces. . . . . . .

The existence of superficial coatings was clearly shown both by the microscope and by photo-micrographs; laboratory experiments indicate decided differences in the behavior of coated and uncoated minerals in the flotation cell. It remained to determine, if possible, the chemical nature and cause of these precipitates or coatings. Two typical cases were taken for this purpose, and were carefully studied. One was the coating present on galena after treatment in a circuit made alkaline by lime, and the other was that formed upon sphalerite in a circuit to which copper sulphate had been added.

In the lime circuit the amount of lime never exceeded three pounds per ton of ore, or a concentration of 0.037 per cent lime in the water. The maximum solubility of lime in water at room temperature is 0.13 per cent, the concentration, then, never exceeded the solubility of lime in water. If all the lime were in soluble form

82 - Gates, J. F., and Jacobsen, L. K., Page 51 of work cited.

it is evident that, since there were no other reagents present, there must have been a reaction between the solute and other substance. A plausible explanation is that a reaction took place between the dissolved lime and the carbon dioxide content of the air beaten into the pulp during the agitation period.

$$CaO + H_2O = Ca(OH)_2 \quad (1)$$

$$Ca(OH)_2 + CO_2 = CaCO_3 + H_2O \quad (2)$$

Calcium carbonate is practically insoluble in water that contains no carbon dioxide. It is probable that this insoluble precipitate, which is deposited on the galena and which so alters its behavior in the flotation cell, is calcium carbonate.

(B). Depression of Pyrite and Blende

Just how lime functions in flotation may be obscure, but the fact that it does function is well established. Lord and Snyder[83] discuss the use of lime as follows:

> . . . . It may be used to neutralize acid salts present in ores or water. It is frequently an aid in clearing the froth of gangue for which purpose it may be best to add it in the cleaning operation. Lime is a strong depressant of pyrite and finds extensive employment for this purpose in the treatment of copper ores. Lime is used with zinc ores to retard pyrite, sometimes being added in the cleaner machines. Unless added in small amounts it depressed lead sulphide.

According to Weinig and Palmer:[84]

> Alkalis are used also as inhibiting or retarding agents, to prevent the flotation of some of the less easily floatable sulphide minerals, such as pyrite and sphalerite. The effect is due, presumably, to the formation of films of basic salts upon the mineral particles.

After stating that lime could be replaced by caustic soda and caustic potash, but because of its much lower cost and greater availability it is the most used alkaline reagent, Weinig and Palmer[85] continue in the same article:

> . . . . . . Lime is extensively used by copper concentrating mills to prevent the flotation of pyrite . . . . .

Flotation reagents may be classified on various bases. Probably the most common and significant basis, however, is that of the apparent or supposed

83 - Lord, Robert, and Snyder, B. M., Notes on the Flotation Process: Bull. of Southwestern Eng. Corp., Los Angeles, Calif., June, 1929, p. 18.
84 - Weinig, A. J., and Palmer, I. A., Page 22 of work cited.
85 - Weinig, A. J., and Palmer, I. A., Page 22 of work cited.

function in flotation, as frothers, collectors, depressants, etc. The classification is not "hard and fast" because a substance that one operator uses and classes as a frother may be used by another as a collector; still another may use it for the combined effect. According to Taggart[86] there are two essential phenomena in flotation.

(a) the degree of water-wetting of solid particles and

(b) the amount and character of froth.

The things that flotation agents do to affect and control these two fundamental phenomena may be grouped into five general classes; the groups of agents, named according to their functions, are: (1) Frothing agents; (2) Collecting Agents; (3) Depressing Agents; (4) Dispersion Agents; (5) Conserving Agents.

Lime plays a part in the last three groups as pointed out by Taggart:[87]

Dispersion agents are substances added to an ore pulp which effect the state of dispersion of the gangue particles and at the same time change the extent to which these particles adsorb at sulphide-particle surfaces . . . . . Common dispersion agents are sulphuric acid, lime, copper sulphate, soda ash, caustic soda, and sodium silicate . . . . .

Depressing agents is the name given to substances that are used to lessen the floatability of one or more of the minerals of the ordinarily-floatable class in a mixture of such minerals, e. g., to depress sphalerite when it occurs with galena, or pyrite when associated with chalcopyrite, and thus make it possible to float the galena and chalcopyrite respectively in concentrates relatively free of sphalerite or pyrite . . . . .

The agents of this class do not all act in the same way. Most of them fall also in the dispersion-agent class; a few react chemically with some sulphides and not with others; and there is some evidence of a third type of action involving preferential deposition of a compound formed by the added agent and some constituent of the ore pulp other than the mineral upon which the salt deposits.

Depressing agents that are also dispersion agents include lime, sulphuric acid, sodium carbonate, sodium bicarbonate, sodium silicate, alkaline cyanide, zinc sulphate, copper sulphate, alum (potassium), and like inorganic substances. Their action is to effect differential adsorption of gangue at the sulphide surfaces.

Continuing, Taggart[88] says:

Conserving agents are substances added to ore pulps to protect the other flotation agents from attack by substances present in the

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86 - Taggart, Arthur F., Pages 841, 844, and 845 of work cited.
87 - Taggart, Arthur F., Page 844 of work cited.
88 - Taggart, Arthur F., Page 846 of work cited.

ore pulp. They are of no particular chemical character and may, in other pulps play some other part in the flotation operation. . . . . Lime used with ores containing free acid, when an alkaline xanthate is used as a collecting agent, is, in part at least, acting as a conserving agent.

Parsons[89] shows that lime was used early in selective flotation:

Lime - This reagent was one of the first to be used in flotation. It requires a time contact with the ore and, if possible, should be added in the grinding mills. It is used to overcome the deleterious effect of soluble salts which are frequently present in ores, but its principal use is as a depressant for iron sulphides in the selective separation of lead and zinc, copper and zinc, and copper and iron sulphides. It also has a marked depressing action on zinc sulphides in their separation from lead and copper. If added in too large amounts it will also prevent the lead from floating. Chalcopyrite will float in a strongly alkaline pulp, but chalcocite and some of the other copper minerals are affected by the least excess. Therefore, lime should not be used in their flotation. . . . .

Farther on in his article Parsons[90] has the following to say regarding the use of lime in the flotation of ores containing lead, zinc, and iron sulphides:

In general, the selective separation of lead and zinc is obtained by the addition to the ore pulp of some reagent that will temporarily deaden the floating properties of the zinc, thus permitting the recovery of a high-grade lead concentrate. The separations are made more difficult by the presence of large amounts of iron sulphides, both pyrite and pyrrhotite, and then some reagent must be used which will permanently deaden these two sulphides so that they will not float with either the lead or zinc. The only reagents used which perform both these functions are sodium cyanide, sodium sulphite and sodium thiosulphate, but the following have been used with some success in temporarily deadening the zinc, zinc sulphate, sodium acid phosphate, and sodium hydrosulphite. The action of zinc sulphate is much more effective when used with sodium cyanide, the two being mixed. . . . . as they enter the flotation pulp. These reagents when used separately seem to have no effect in preventing the iron sulphides from floating.

These modifying reagents are used in an alkaline pulp, with either soda ash or lime. Lime has a permanent deadening effect on pyrite and pyrrhotite, and also affects galena in the same way but to a lesser degree . . . By the use of copper sulphate, the action of these modifying reagents is destroyed, and the flotative properties of the zinc minerals are revived . . . .

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89 - Parsons, C. S., Selective Flotation: Eng. and Min. Jour., vol. 123, May 7, 1927, p. 758.

90 - Parsons, C. S., Page 759 of work cited.

Weinig and Palmer[91] add the following concerning the use of lime in the lead-zinc separation:

. . . . If present in large quantities, lime practically inhibits the flotation of all sulphide minerals. For this reason lime is not used in lead-zinc separations where the mill water is used over again. The lime that is added to inhibit the flotation of pyrite in the zinc-iron separation would interfere with the flotation of galena in the head of the circuit. In such cases sodium carbonate is used to produce the alkalinity, because this reagent does not seriously hinder the flotation of the lead mineral . . . .

Robie[92] also writes of the use of lime in the lead-zinc circuit:

Fresh water is used in the mill circuit, as the reagents used in floating the zinc make the water from the settling ponds unadaptable for lead flotation. Soda ash is used instead of lime, because it does not raise so much iron, and the pyrite is very readily floated when finely ground. Caustic soda gives too stiff and heavy a froth . . . . . .

Diamond[93] writing on the same subject says:

Soda ash is used in the three mills as a conditioning and alkaline reagent. Caustic soda and lime have been found unsuitable in our operations, to date. The quantity of soda necessary for good work varies with the character of the ore being treated, our range being from 1 to 10 lbs. per ton. The demand is usually constant with each ore. Tailing solution can generally be returned for soda and heat saving, together with thickener overflows and filtrates, into the circuit without noticeable fouling.

Parsons[94] in writing about the separation of intimately mixed chalcopyrites, sphalerite, and iron sulphides says:

This type of ore probably presents the most difficult of all selective flotation problems. The use of cyanide temperarily to deaden the zinc and permanently to deaden the iron sulphides seems essential. However, laboratory separations have been obtained by floating the copper in an alkaline pulp by the addition of lime. For the flotation of the zinc the copper tailing was dewatered to eliminate a part of the lime and the pulp then brought up to the required density by addition of fresh water. At this point soda ash was added and the zinc floated in a soda ash pulp, copper sulphate being used to help reactivate the zinc. . . . .

91 - Weinig, A. J., and Palmer, I. A., The Trend of Flotation: Quart. Colorado Sch. of Mines, vol. 24, No. 4, 2d ed., Apr., 1928.

92 - Robie, Edward Hodges, Selective Lead Zinc Flotation at Sunnyside: Eng. and Min. Jour., vol. 121, May 8, 1926, p. 760.

93 - Diamond, R. W., Flotation Reagents at the Sullivan Mill: Min. and Met., vol. 8, Aug., 1927, p. 336.

94 - Parsons, C. S. Page 761 of work cited.

In the following statement Johnson[95] disagrees with Parsons on the action of lime as a depressant for sphalerite:

> Mr. Parsons speaks of lime as an active depressant of sphalerite. On Chief ores this does not apply; lime actually increased the floatability of sphalerite if added in amounts up to 8 or 9 lbs. per ton of ore. In practice the addition of lime, as the hydrate, ranges from 2 to 4 lbs. per ton; additions in excess of 4 lbs. per ton tend to coagulate the gangue to such an extent that a high 'insoluble' content appears in the zinc concentrate. . . . .

Booth[96] adds the following comment to this discussion:

> I believe that generally the use of lime has very little if any activating quality for the flotation of sphalerite. It is likely that when lime is used it sometimes give rise to the erroneous impression that sphalerite has been activated, whereas in reality the other minerals (pyrite in particular and possibly galena) have been depressed. I also find when using lime as a reagent in sphalerite flotation, especially large quantities of lime--and 8 or 9 lbs. per ton is a large quantity--that generally more copper sulphate is required for the actual activating than when using soda ash or other alkalies . . . .
>
> As regards the use of soda ash, I have tested a number of ores in which pyrite flotation was improved by its use. Usually, however, its benefits are more apparent than real. It usually gives a better looking pyrite mineral froth, and occasionally the assays show a better recovery of iron.
>
> In the testing of ores containing large quantities of pyrrhotite, I have found that copper sulphate has a decided activating effect on the pyrrhotite, especially if some means are not taken for depressing the mineral previous to the addition of the copper sulphate.

Another record of the use of lime in flotation is given by MacDonald[97]:

> Selective flotation, in this sense, between chalcopyrite and pyrite, has been achieved in the mill of the Moctezuma Copper Co. at Nacozari, Sonora, Mexico, to the extent that over 90 per cent of the iron combined as pyrite in the original mill feed is now being rejected, whereas the percentage of copper recovered remains almost at the high level obtained when but 25 per cent of the pyrite iron was being rejected . . . .
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> Dry lime is fed mechanically onto the ore conveyors leading from the fine-ore bins to the primary mills . . . . . .

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95 - Johnson, Homar L., The Action of Certain Reagents in Selective Flotation: Eng. and Min. Jour., vol. 123, June 4, 1927, p. 931.
96 - Booth, Lionel E., Effects of Bichromate in Flotation at the Sullivan Mill: Eng. and Min. Jour., vol. 124, July 23, 1927, p. 141.
97 - MacDonald, Wm. T., Pages 445-454 of work cited.

Best flotation results are obtained when the tailing pulp shows a lime content equivalent to 0.15 to 0.35 lbs. of calcium oxide per ton of clear solution, depending somewhat upon the quality of the lime used. When the alkalinity falls below this minimum, high flotation tailing usually ensues, whereas an unduly high alkalinity changes the character of the froth unfavorably and involves a waste of reagent. . . . .

When a good quality of lime is available, the consumption averages from 5 to 6 lbs. per ton of flotation feed . . . .

The functions of lime in flotation include the neutralizing of natural acids, precipitation of possibly detrimental soluble salts, prevention of oxidation of the sulphides, coagulation and deflocculation, and modification of physical properties of the electrolyte.

Locke[98] writes of the separation of lead, copper, and iron as follows:

> One section of the new Utah Apex mill handles lead-copper-iron sulphides. The lead and copper are first floated together by the use of xanthate, thus eliminating the iron with the tailings. The lead-copper froth is next conditioned with 1 lb. lime and 0.5 lb. sodium sulphide per ton, thus deadening the copper and enabling the lead to be floated from it in the final separation step.

Geiser[99] mentions the use of lime in differential flotation and also the questionable advantage of lime in a lead-zinc circuit:

> Lime is the most important alkaline reagent employed in flotation today; it is generally cheap and abundant and as its positive action in depressing pyrite is becoming better understood, more plants are adopting it. The rejection of barren pyrite in the concentration of copper ores by the use of lime and the T-T mixture or xanthate has made profitable the treatment of lower grade ores and has enabled mines to keep operating which would otherwise be forced to discontinue during the present low price of copper. It is generally used on copper ores, but frequently excellent results can be obtained on lead-zinc ores. The amount to use will vary in individual cases, but is generally critical, probably not over one pound per ton of solution being sufficient. . . . . . . High alkalinity will depress galena, but has not so harmful an effect on the sphalerite, though more xanthate will be needed . . . .
>
> In the selective flotation of chalcopyrite and nickel minerals in the presence of pyrrhotite the essential requirements are the same as in eliminating pyrite--namely, pulp made alkaline by lime or soda ash, and the new flotation agents. The chalcopyrite floats with more ease than the nickel minerals.

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98 - Locke, Chas. E., Flotation Dominates Ore-Dressing Progress: Eng. and Min. Jour., vol. 123, Jan. 22, 1927, p. 151.

99 - Geiser, H. S., Flotation Reagents and Practice: Eng. and Min. Jour., vol. 123, May 21, 1927, pp. 843-44.

(C). Depression of Gold and Copper Metallics

In connection with the flotation of metallic copper Fahrenwald[100] says that proper alkalinity control is important and that lime may be used, although he advocates sodium dibasic phosphate:

. . . . On some samples, tailings carrying as little as 0.06 per cent copper were repeatedly obtained.

Xanthate in a slightly alkaline pulp in conjunction with a variety of oils was the reagent combination used. Lime or sodium carbonate or sodium dibasic phosphate in small quantities gave satisfactory alkalinity. Sodium dibasic phosphate was the best of these substances; a given quantity was satisfactory, regardless of the mill from which the table-feed samples were taken. Lime or soda ash called for closer regulation. The mill water was slightly less than pH = 7 and a pH of 7.5 to 8 was required . . . .

Regarding gold and silver ores von Bernewitz[101] says in conclusion:

. . . . . More interest is being taken in the flotation of gold--and silver--bearing ores, and the scope of the process is widening.

Substantiating this statement is a current press report[102] which shows how flotation is competing with cyanidation:

McIntyre Porcupine Mines, second largest gold producer in the Porcupine district of Ontario, will build a 2000-ton flotation plant to replace the present 1500-ton cyanidation mill . . . . .

The importance of lime in this new flotation is brought out by Parsons[103] in discussing differential flotation of complex ores with and without gold:

Successful separation of chalcopyrite from the iron sulphides, both pyrrhotite and pyrite requires the solution of two problems. The first problem is to free the copper mineral from the iron . . . . . The second problem is to prevent the iron from floating with the copper. With chalcopyrite ore containing no gold this can generally be done by maintaining a pulp strongly alkaline with lime. Lime has a permanent deadening effect on pyrite and pyrrhotite and very high ratios of concentration can be obtained.

As a general rule, copper ores containing gold present an entirely different problem, that of recovering the gold in the copper

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100 - Fahrenwald, A. W., Xanthate and Pure Oil Float Native Copper in Amygdaloid Ores: Eng. and Min. Jour., vol. 126, July 14, 1928, p. 59.
101 - von Bernewitz, M. W., Flotation in the Treatment of Gold and Silver Ores; A Review: Eng. and Min. Jour., vol. 124, Oct. 15, 1927, p. 657.
102 - Engineering and Mining Journal, vol. 129, June 23, 1930, p. 631.
103 - Parsons, C. S. Page 760 of work cited.

concentrate. The use of lime to retard the iron, except in rare instances, tends to throw the greater part of the gold with the iron tailing. Soda ash or caustic soda is used because it has not the detrimental effect on gold, but unfortunately it does not deaden the pyrite. Its effect is sufficient to keep the pyrrhotite from floating, but pyrite floats readily. Soda disperses or deflocculates gangue slime, whereas lime coagulates it. However, soda also seems to tend to coagulate the sulphides.

Sodium cyanide added to a pulp, the alkalinity of which is maintained with soda ash, can be used permanently to deaden at least a part of the pyrite, but it also often has a modifying effect on the chalcopyrite. The chalcopyrite can be reactivated by the addition of a limited amount of copper sulphate. By the use of cyanide under these conditions it is often possible to obtain a higher ratio of concentration without lowering the recovery of gold.

In connection with what Parsons says concerning the gold in flotation, Thomas, Christmann, and Gifford[104] write:

> The third example is an ore from Southwestern United States assaying 1.87% copper and 14.8% iron. The ore contains some metallic copper and the comparison of lime and cyanide is of special interest for this reason. It has been observed on several occasions that lime is a powerful depressant for metallics, whether gold or copper, and this is well brought out by the data . . . . . where the lowering of the recovery caused by the use of lime is doubtless attributable, in part at least, to failure to recover metallic copper . . . .

To this, von Bernewitz[105] cites a case where: "Flotation in a high-alkaline cyanide solution was not a success . . . .

CONCLUSION

Thus, lime, which was one of the first chemicals to be used by man, is still one of the most important industrial reagents. Its use as a purifier in the treatment of water has been well established; and in milling it has served for many years in cyanidation and amalgamation. In the last five years it has become the most generally used flotation reagent. About 2.5 per cent of the nation's output is consumed by the flotation process alone.

Since there are many varieties of lime and each has its advantages and disadvantages, the operator who has occasion to use it is confronted with the question of a choice. Such a choice must be guided by both technical and economic considerations. Consequently, no general recommendation is warranted, for what is good practice at one mill may be questionable elsewhere.

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104 - Thomas, Christmann, and Gifford, Page 6 of work cited.
105 - von Bernewitz, M. W., Page 657 of work cited.

Limes vary greatly in composition and properties; but the multiplicity of its uses gives a field of application for practically every variety. On the other hand, the types suitable for a particular operation may be very limited. There may be one variety which is superior, but a less desirable lime may be chosen because of its cheapness. An economic balance between advantages and disadvantages must be established, and that must be by the operator.

In general, for flotation, a high-calcium hydrated lime is probably the most favored. It is active as a neutralizer of acid, imparts alkalinity, is a purifier and a depressor, and is easily handled. Its chief disadvantage is its cost. Economically, leaner limes might be justified despite their inefficiency.

The chemical, lime (CaO), is the most desirable constituent of a "lime," and hence the "available lime" content is often an indication of the value of the material. Quicklime is CaO (plus impurities) and therefore might be considered better than the hydrate ($CaOH$) or those limes which contain magnesia (MgO) and other impurities. However, the hydrate is easier and safer to handle, and in some cases dolomitic limes are not disadvantageous. For neutralizing acid, dolomitic lime is very effective, but it is not efficient in imparting alkalinity because the magnesium hydrate is insoluble. If free acid is to be neutralized, unburned limestone may be used, and subsequent treatment may be done with lime. Most of the impurities in lime act only as fillers and are not harmful in themselves; the chief active constituent is the soluble calcium oxide or hydroxide.

In choosing a lime, an operator must consider the purpose of adding lime, the properties of the particular lime chosen, and the value of the limes with respect to their costs. Since there are no hard and fast rules to govern the selection, operators differ in opinion and final choice of lime.

The method of using lime is also subject to personal opinion and plant operation. In general, lime for water purification and as a reagent in flotation should be added to the circuit early enough to allow it to be most effective in its reactions.

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Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

EXPLOSIONS IN TENNESSEE COAL MINES

BY

H. B. Humphrey

I. C. 6424
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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

EXPLOSIONS IN TENNESSEE COAL MINES 1/

By H. B. Humphrey 2/

INTRODUCTION

In the 39 years from 1891 to 1929, gas and dust explosions in the coal mines of Tennessee caused 413 deaths, or one-third of the fatalities in the mines for that period. Inasmuch as explosions caused but about 15 per cent of mine fatalities for the whole United States for this same period it is evident that the Tennessee coal mining record is by no means good, as far as explosion fatalities are concerned. Since 1926, Tennessee has had a clear record as to fatalities from explosions; this fact may be credited very largely to the improved conditions and safer practices in Tennessee's coal mines which have been brought about by the work of the State Inspection Department and of the Southern Appalachian Coal Operators' Association directed toward adoption of safety precautions and practices most of which are advocated by the United States Bureau of Mines. Chief among these improvements are the replacement of black powder and dynamite by permissible explosives, improved ventilation, and the installation of rigid methods of inspection for gas. With the present tendency for operations to pass into the hands of responsible and informed management, and with unrelaxed care and inspection, the good record of the past several years should be maintained.

ACKNOWLEDGMENTS

Material for this paper was obtained through the offices of A. W. Evans, Chief Mine Inspector, Nashville, Tenn., and from United States Bureau of Mines Bulletins 115, 196, and 293 on coal mine fatalities in the United States. Reports of Bureau of Mines engineers on explosions were consulted and F. E. Cash, district engineer, assisted with information as to conditions and practices in Tennessee mines.

DISCUSSION OF DISASTERS

Table 1 is a list of the fatalities from gas and dust explosions from 1891 through 1929, as obtained from the reports of the State inspector, the reports of bureau engineers and from the bulletins on coal mine fatalities published by the Bureau of Mines. Except for cases occurring during the war

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6424."

2/ Assistant engineer, U. S. Bureau of Mines, Mine Safety Station, Birmingham, Ala.

Table 1.- Gas and Dust Fatalities in Tennessee

Date	Company and mine	Type	Origin	Class mine	Explosive used	Killed
1891 Jan.	Coal Creek Coal Co. Thistle mine	Gas	Open light	B	Black Powder	1
Oct.	Roane Iron Co. Rockwood mine	do.	do.	A	do.	1
1895 Dec. 20	Dayton Coal & Iron Co. Nelson mine	Gas and dust	Blown-out shot	B	do.	28
1898-1899		Gas do.	Unknown do.			1 1
1901 May 27	Dayton Coal & Iron Co. Richland mine	Dust	Blown-out shot keg of powder	C	B. P.	20
1902 Mar. 31	Dayton Coal & Iron Co. Nelson mine	Gas and dust	Blown-out shot	B	B. P.	16
1902 May 19	Coal Creek Coal Co. Fraterville #1 mine	do.	Open light	B	B. P.	184
1904		Gas	Unknown			1
1904 Jan. 5	Poplar Creek Coal Co. Big Mountain mine	Dust	Blown-out shot	A	B. P.	3
1906	Roane Iron Co. Rockwood mine	Gas	Open light	A	Dynamite and B. P.	1
1907 July 14	Bear Wallow Coal Co. Bear Wallow mine	do.	do.	A	Dynamite and B. P.	1
1907 Nov. 1	Windrock Coal & Coke Co. Windrock mine	do.	Blown-out shot	C	Dynamite and B. P.	1
1907 Dec. 23	Bowling Coal Co. Bowling mine	do.	Open light	C	Dynamite and B. P.	1
1909 Sept. 21	LaFollette C. I. & R. R. Co. Rex No. 2 mine	Gas and dust	Blown-out shot	A	Dynamite and B. P.	1
1910 Dec. 23	W. Phillips Paint Rock No. 3 mine	Dust	do.	C	Dynamite and B. P.	3
1911 Dec. 9	Knoxville Iron Co. Cross Mountain No.1 mine	Gas and dust	Open light	B	B. P.	84
1913	State of Tennessee State mine, Petros	Fire and gas	Fire, origin ?	A	Permissible B. P.	1

Table 1.- Gas and dust fatalities in Tennessee (cont.)

Date	Company and mine	Type	Origin	Class mine	Explosive used	Killed
1914 Aug. 18	Durham Coal & Iron Co. Soddy 1-2	Dust	Blown-out shot	C	B. P.	1
1915 Apr.		CO_2 Gas	Suffocation			1
1917 Oct.		Gas	Open light			1
1917 Dec. 20	Barbour Coal & Coke Co. Nemo mine	Dust	Blown-out shot	C	B. P.	11
1919 Mar. 9	Bills Branch Coal Co. Bills Branch No. 1 mine	Gas	Open light	D		1
1919 Dec. 24	Paint Rock Coal Corp. Paint Rock No. 1 mine	Dust	Blown-out shot	B	B. P.	1
1920 Feb. 3	Pemberton-Hibbler Min. Co. Hibbler No. 1 mine	do.	do.	C	B. P.	1
1922 Nov. 20	Highland Coal & Lbr. Co. Highland No. 3 mine	Gas and dust	Blown-out shot	B	B. P. and Dynamite	1
1923 Feb. 26	East Laurel Mining Co. East Laurel mine	CO_2 gas	Suffocation	D	B. P.	1
1923 Dec. 10	Windrock Coal & Coke Co. Windrock mine	Dust	Blown-out shot	C	B. P.	2
1924 Dec. 31	Vaspar Coal Mining Co. Vaspar mine	Gas	Open light	A	B. P.	3
1925 Jan. 5	Roane Iron Co. Rockwood mine	do.	do.	A	Permissible[1]	2
1925 July 23	Roane Iron Co. Rockwood mine	Gas and dust	Open light	A	Permissible	10
1925 Dec. 9	Jackson-Laxton Coal Co. Hagemeyer mine	Gas	Ignition open light	C		1
1926 Oct. 4	Roane Iron Co. Rockwood mine	Gas	Open light	A	Permissible	27

1 Overloaded and tamped with coal-dust.

years when no State reports were published and in 1898, 1899, and 1904, the causes of injury are given, with the class of mine and the number killed. According to the State mining laws, Class A mines are those liberating methane; Class B mines are those dangerously dry and dusty; Class C mines include all mines not in Class A or B, employing over 20 men and 3 mules; Class D includes those employing less men than Class C mines.

In classifying mines the presence of methane is determined by detection with a flame safety lamp.

Of the 413 deaths, Class A mines had 50, Class B had 315, Class C had 41, Class D had 2, and 5 occurred in mines, the classification of which is not known. In 1914 there were 20 Class A mines listed and 7 Class B; in 1926 there were 14 of Class A and 2 of Class B and in 1927 there were 7 of Class A and 4 of Class B. This predominance of Class B is due to bad disasters in mines which were in Class B but which also liberated gas. Of all the explosions listed, 10 were in Class A; 7 in Class B; 9 in Class C; 2 in Class D; and 5 in unlisted mines.

Table 2 gives those explosions classed as major disasters in which five or more men were killed.

Table 2. - Major disasters

Date	Mine	Class	Type	Origin	Explosive used	Killed
1895	Nelson	B	Gas and dust	Blown-out shot	Black powder	28
1901	Richland	C	do.	do.	do.	20
1902	Nelson	B	do.	do.	do.	16
1902	Fraterville	B	do.	Open light	do.	184
1911	Cross Mt.	B	do.	do.	do.	84
1917	Nemo	C	Dust	Blown-out shot	do.	11
1925	Rockwood	A	Gas and dust	Open light	Permissible	10
1926	do.	A	do.	do.	do.	27
Total	--	--	--	--	--	380

The one class A mine in this list had two disasters with 37 killed; this mine is now closed, the equipment has been withdrawn, and operations are suspended indefinitely. Three Class B mines had four explosions with 312 deaths, and the other two explosions were in Class C mines with 31 killed; dust was a factor in every case in propagating the explosion. The two greatest disasters were in Class B mines and were set off by open lights in accumulations of gas. Of the eight major disasters four were set off by open lights and four by blown-out shots, where black blasting powder or black blasting powder and dynamite were used.

Table 3.- Explosions classified by type

Type	No.	Fatalities	Percentage	
			Explosions	Fatalities
Gas	17	20	52	5
Dust	8	42	24	10
Gas and dust	8	351	24	85
Total	33	413	100	100

Seventeen of the total explosions, or over 50 per cent, were caused by the ignition of gas alone and resulted in 20 deaths; eight disasters with 42 deaths were due to explosion of suspended dust, and the remaining eight were combined gas and dust explosions resulting in 351 deaths.

Of the 17 gas ignitions, 15 caused single fatalities, one resulted in three deaths, and the other in two deaths. Of these, 6 occurred in Class A mines, 1 in Class B mines, 3 in Class C, 2 in Class D, and 5 in mines not listed. Less than half occurred in mines classed as gassy, and more than half were in Classes C and D.

Dust explosions occurred in one Class B mine with 1 fatality and seven Class C mines with from 1 to 20 fatalities in each.

Three gas and dust explosions occurred in Class A mines and five in Class B mines, all except two being major disasters; this shows that the mines having the worst disasters were recognized and labeled as dangerous. There were 351 men killed in these eight explosions, two of which caused single fatalities, and the rest resulting in from 10 to 184 deaths per explosion. These occurrences formed less than one-fourth of those recorded but they had 85 per cent of the fatalities.

Origin of Explosions

Table 4. - Explosions classified by origin

Origin	All explosions		Per cent		Major disasters		Per cent	
	Number	Fatalities	Explosions	Fatalities	Number	Fatalities	Explosions	Fatalities
Open light-------	14	318	42	77	4	305	50	80
Blown-out shot	13	89	40	21	4	75	50	20
Uncertain -------	3	3	9	1	-	--	--	--
Miscellaneous	3	3	9	1	-	--	--	--
Total	33	413	100	100	8	380	100	100

Table 4 gives data concerning the origin, number, and the fatalities resulting from all explosions and from major disasters only.

The peculiar feature of this district is that essentially half of the explosions, both major and minor, have been caused by open lights and about half by blown-out shots, the latter being evidence of past unsafe shooting practice. Open lights have been the chief cause of ignition since 1924. No proved cases of electrical origin have occurred; this may be explained by the fact that there are few gassy mines and there is relatively little use of electrical equipment. However, the danger from electricity will increase as its use is extended, making it needful to keep open type electrically operated machines from gassy workings and to keep gas and suspended coal-dust from contact with electrical installations by the use of adequate and controlled ventilation, rock-dust, and water. Some cases of suffocation have occurred, probably chargeable to poor ventilation. In both major and minor explosions, open lights caused about 80 per cent of the fatalities and blown-out shots 20 per cent; this figure is not of as much significance as might be given to it since it is largely a result of the death of 184 men at one time in 1902 in an explosion set off by an open light.

Small "local ignitions" should be guarded against by the use of closed lights in any mines where accumulations of gas may occur, and by proper inspection and ventilation to prevent such accumulations. These "local ignitions" are potential disasters, much easier to provent than to control.

CONCLUSIONS

Experience of several years has shown that explosions are preventable in Tennessee coal mines as well as in coal mines everywhere else. The good record since 1926 can be maintained in the future by frequent and rigid inspection, efficient supervision, and the proper use of ventilation, rock-dust, water, and of permissible equipment and explosives in mines which are or are likely to be gassy or dusty.

The adoption and extension of the use of electrical equipment must be accompanied by such safeguards as will prevent the ignition of gas accumulations or suspended coal-dust by possible arcing or short circuiting.

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FACTORS INVOLVED IN THE HEAP LEACHING OF COPPER ORES

BY

John D. Sullivan

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

FACTORS INVOLVED IN THE HEAP LEACHING OF COPPER ORES[1]

By John D. Sullivan[2]

INTRODUCTION

The recovery of copper by leaching from relatively low grade ores containing both oxidized and sulphide minerals is rapidly gaining in commercial importance. There is an economic limit below which certain ores can not be treated by concentration methods, and material with values below this limit must be discarded, possibly to be treated later, or the copper must be extracted by wet processes.

Heap leaching has been carried out by the Phelps Dodge Corporation, Copper Queen Branch, for several years, and to-day its plant is a large producing unit. The Ohio Copper Co. at Bingham, Utah, has been leaching in place since 1919. Many other leaching operations are also being carried out in various parts of the United States and in foreign countries. The plant at Rio Tinto, Spain, may be considered the father of heap leaching.

In heap leaching the ore is usually heaped as it comes from the mine without any breaking or crushing treatment. Frequently pieces of rock 5 or 6 feet across a face are found in the heaps. Solution is added to the ore at the surface and seeps downward by gravity and is collected in a solution sump. Solution is usually added to one section of a heap and, after a certain quantity has been added, the solution is added to another section. Sometimes weeks or even months elapse between additions of solution to a given section.

Four factors are essential to the successful leaching of any copper-bearing ore: (1) A solution that will attack the copper minerals must get into the body of the ore particles, (2) the copper minerals must be dissolved by the solution, (3) the solution containing the copper must find its way out of the voids into the main solution stream, and (4) the copper in solution must be recovered by some means of precipitation.

The United States Bureau of Mines at its Southwest Experiment Station, in cooperation with the department of mining and metallurgy, University of Arizona, has undertaken a study of these fundamental factors involved in the

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6425."

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leaching of copper ores. So far, the study has been limited to the first three factors, which are those involved in the leaching operation itself. In leaching a given ore the rate of extraction is not instantaneous but goes on slowly, and the factors of penetration, dissolution, and diffusion go on siumltaneously and not in successive steps. However, these steps can be studied only by segregating them so as to have only one factor entering at a time. The results thus obtained lend themselves to comparisons that can be used in commercial practice.

This paper presents a resume of the results obtained in the heap-leaching studies made at the Southwest Experiment Station, and the general conclusions which have been drawn.

Former papers have described in detail the experimental procedure and the results obtained in the study of: (1) The ingress of solutions into ores during leaching,[3,4] (2) the dissolution of various copper minerals,[5,6,7,8,9] and (3) the removal of soluble copper from leached ores.[10,11,12]

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3 - Sullivan, J. D., Keck, W. E., and Oldright, G. L., Factors Governing the Entry of Solutions into Ores during Leaching; Tech. Paper 441, Bureau of Mines, 1929, 38 pp.

4 - Sullivan, J. D., and Ostrea, Enrique, Factors Governing the Entry of Solutions into Ores during Leaching II: Tech. Paper 498. Bureau of Mines; in press.

5 - Sullivan, J. D., Dissolution of Various Oxidized Copper Minerals: Rept. of Investigation 2934, Bureau of Mines, 1929, 9 pp.

6 - Sullivan, J. D., and Oldright, G. L., The Dissolution of Cuprite in Sulphuric Acid and in Ferric Sulphate Solutions: Rept. of Investigation 2967, Bureau of Mines, 1929, 9 pp.

7 - Sullivan, J. D., Chemistry of Leaching Chalcocite: Tech. Paper 473, Bureau of Mines, 1930, 24 pp.

8 - Sullivan, J. D., Chemistry of Leaching Bornite: Tech. Paper 486, Bureau of Mines, 1931, 20 pp.

9 - Sullivan, J. D., Chemistry of Leaching Covellite: Tech. Paper 487, Bureau of Mines, 1930, 18 pp.

10- Sullivan, J. D., and Sweet, A. J., Factors Governing Removal of Soluble Copper from Leached Ores: Tech. Paper 453, Bureau of Mines, 1929, 26 pp.

11- Guggenheim, M., and Sullivan, J. D., Acceleration of Extraction of Soluble Copper from Leached Ores: Tech. Paper 472, Bureau of Mines, 1930, 30 pp.

12- Sullivan, J. D., and Bayard, K. O., Extraction of Soluble Copper from Ores in Leaching by Percolation: Rept. of Investigations, 3073, Bureau of Mines, 1931, 21 pp.

INGRESS OF SOLUTIONS INTO ORES

In leaching an ore, the solution must first permeate the ore in order to come into contact with the copper-bearing minerals. There are, in general, two classes of voids in rocks: (1) Fractures and cleavage planes and (2) pores of capillary size or smaller. The crevices and cleavage planes are usually of such a nature that they are open along one side or at both ends. For a solution to enter these openings it must mechanically push the gas out of the voids. As these openings are not in general in a horizontal position, there may be a larger hydrostatic head at one end of the opening than at the other. It is not particularly difficult for solutions to enter large voids. In the second class of voids, the pores may extend through the rock or they may be connected by a network to one another so that they are open at both ends, but pores open at one end only are also present. Even in small pores open at both ends, if solution penetrates simultaneously from both ends, the pores act as though closed at one end. In the case of a pore closed at one end, solution can penetrate only by replacing the gas within the pore. A solution will not penetrate unless the pressure within the pore is less or equal to the outside pressure. The gas in the closed part of the pore must either dissolve in the penetrating liquid or diffuse through it, if penetration takes place.

The movement of liquids in rocks through supercapillary openings (circular openings greater than 0.508 mm. in diameter, or sheet openings greater than 0.254 mm. between walls) is probably governed by the ordinary laws of hydrostatics. Poiseuille's law probably governs the movement of liquids in capillary openings (circular openings 0.0002 to 0.508 mm. in diameter, or sheet openings 0.0001 to 0.254 mm. between walls). Flow through subcapillary openings would be infinitely slow, as the attraction of solid molecules extends from wall to wall, and no liquid would be free to move, as it is attracted to the walls as a film.

A description has been given elsewhere[13] of the apparatus and procedure used for determining: (1) The rate and volume of penetration of solutions into ores, (2) the total volume of voids within particles of ores, and (3) the density of ores.

Table 1 shows the rate at which water penetrated into various sizes of a typical porphyry ore of the Southwest. The data show that the time needed for a given percentage penetration increases markedly with increases in size of particles.

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13 - Sullivan, J. D., Oldright, G. L., and Keck, W. E., Method for Measuring Voids in Porous Materials: Rept. of Investigations 3047, Bureau of Mines, 1930, 8 pp.

Table 1. - Rate of penetration of distilled water into various sizes of a porphyry ore; temperature, 30° C.

Time, hours	Voids filled, per cent			
	-1 3/4-inch pieces	2-inch pieces	3-inch pieces	4-inch pieces
1/60	66.1	31.4	20.4	16.8
1/30	69.3	37.2	26.5	21.8
1/10	75.7	45.7	34.2	28.1
1/4	80.5	54.2	44.7	33.7
1/2	83.6	59.5	53.8	37.5
1	86.4	63.6	62.0	42.0
2	89.2	68.4	69.2	46.9
3	89.8	71.6	70.4	50.2
5	91.0	75.0	71.5	53.3
23½	95.3	80.5	75.7	62.2
48	96.8	82.0	77.2	66.3
84	98.1	83.0	78.5	69.1
108	98.1	83.5	79.4	71.2
336	---	86.0	83.5	81.3
360	---	---	83.7	82.0
792	---	---	89.4	91.4
888	---	---	---	92.2

Surface Tension.--The general opinion has been that solutions entered the pores of rocks by capillarity. If capillarity is the governing factor, then by changing the surface tension of the penetrating liquid, the rate of entry of solution should also be changed. Tests were made in which the surface tension of water was lowered from approximately 75 dynes to about 25 dynes per centimeter[14] by adding enough sodium oleate to make a saturated solution, but the rate of entry of solution into the ore was practically identical with that of pure water.

Solubility of Gas in Voids.--As surface tension has little or no effect upon the rate of penetration of solution into ores, the rate must be governed primarily by some other factor, which is indicated to be the solubility in the penetrating solution of the gas or gases within the voids of the ore. The solubility of sulphur dioxide[15] in water is 3,937 cubic centimeters in 100 cubic centimeters of water at 20°C., whereas the solubility of air[16] is 1.8 cubic centimeters in 100 cubic centimeters of water at the same temperature. Data in

14 - Harkins, W. D., Davies, E. C. H., Clark, G. L., Surface Energy, VI: Jour. Am. Chem. Soc., vol. 39, 1917, p. 587.

15 - Seidel, A., Solubilities of Inorganic and Organic Substances: New York, 1917, p. 329.

16 - Seidel, A., Page 10 of work cited.

Table 2 show the rate at which distilled water penetrated into various sizes of a typical porphyry ore which had been evacuated and the voids filled with sulphur dioxide. The ore was the same as that used in the tests summarized in Table 1. When the voids were filled with sulphur dioxide, water penetrated more rapidly, especially during the early part of the impregnation. Not only was the rate of penetration faster, but the total volume of penetration was also greater.

Table 2.--Rate of penetration of distilled water into various sizes of a porphyry ore the voids of which were filled with sulphur dioxide; temperature, 30° C.

Time, hours	Voids filled, per cent			
	-1+3/4-inch pieces	2-inch pieces	3-inch pieces	4-inch pieces
1/60	69.5	45.0	--	43.3
1/30	78.0	52.0	50.2	45.8
1/10	84.6	65.0	66.4	50.1
1/4	90.0	80.0	79.4	55.4
1/2	95.3	86.3	85.2	58.2
1	95.8	89.2	88.5	62.6
2	96.4	92.0	90.6	66.4
3	97.1	93.4	91.0	68.5
5	97.7	94.0	92.2	71.0
$23\frac{1}{2}$	98.8	95.5	94.3	77.2
48	99.1	96.7	95.4	79.8
84	99.7	97.7	96.0	82.0
108	100.0	98.8	96.2	83.0
336	--	100.0	97.9	88.4
360	--	--	98.1	89.1
792	--	--	100.0	98.4
888	--	--	--	100.0

Temperature.--Measurements made at 2 to 3.5° C. and at 35° C. showed that the rate of penetration was more rapid at the lower temperature. For a given ore, 95 per cent of the total penetration that took place was attained in 40 hours at 2 to 3.5° C., whereas 50 hours was required at 35° C. As the solubility of gases in water increases with a decrease in temperature, the solution might be expected to penetrate at a faster rate at the lower temperature.

Various Solutions.--There is surprisingly little difference in the rate of penetration of various kinds of solutions into rocks; 5 per cent copper sulphate, 2 per cent sulphuric acid, 2 per cent copper sulphate or ferrous sulphate plus sulphuric acid, 2 per cent ferric sulphate, and distilled water have very nearly the same rates of penetration.

Manner of Penetration.--It has been found that in most ores the solution enters along fractures and cleavage planes and that from these larger fractures it gradually seeps into the rest of the rock. The first penetration

takes place very rapidly along these crevices and fractures, and the small voids are filled more slowly from these points of initial penetration. Photographs demonstrating this phenomenon can be found elsewhere.[17]

DISSOLUTION OF VARIOUS COPPER MINERALS

Oxidized Minerals.--The oxidized minerals of copper—azurite, malachite, chrysocolla, and tenorite—are soluble in sulphuric acid and in ferric sulphate solutions. Approximately 100 per cent of the total copper is soluble in one hour at 35° C. in sulphuric acid or in acidified ferric sulphate solutions when the size of mineral is minus 100 plus 200 mesh. In 1 or 2 per cent neutral ferric sulphate the dissolution is slower, but in nearly every instance the mineral is completely dissolved in 24 hours or less. The rate of dissolution in 5 per cent ferric sulphate is about the same as in sulphuric acid or in acidified ferric sulphate.

Cuprite.--When cuprite is leached with sulphuric acid in the presence of excess atmospheric oxygen, approximately 100 per cent of the copper in pieces up to 4-mesh in size is converted into the soluble sulphate in 20 days at 35° C. With a minus 3 plus 4 mesh product, approximately 94 per cent is rendered soluble in the same time. For pieces 100 mesh or smaller in size, 24 hours is sufficient for complete dissolution. When cuprite is treated with acidified ferric sulphate, particles up to 3 mesh in size are completely dissolved in 8 days and in 3 days 99 per cent is dissolved. For 100-mesh or smaller particles, one hour is sufficient for complete dissolution. When cuprite is leached with sulphuric acid in the absence of oxygen (as, for example, in closed bottles in an atmosphere of nitrogen) only one-half of the copper is rendered soluble, and one-half of it remains as metallic copper, according to the reaction:

$$Cu_2O + H_2SO_4 = CuSO_4 + Cu + H_2O.$$

When cuprite is leached, either in the presence or in the absence of oxygen, a layer of metallic copper forms practically as soon as the mineral comes in contact with sulphuric acid. This metallic copper forms a difficulty permeable layer on the surface of the particles that slows down the dissolution. The metallic copper may be converted to copper sulphate by the aid of an oxidizer. Atmospheric oxygen is a fairly good oxidizer but ferric sulphate is a much better one. On particles 100 mesh or smaller in size this metallic coating of copper does not markedly hinder the rate of dissolution, but it is very harmful for larger sizes.

Chalcocite.--Chalcocite is the principal sulphide mineral encountered in leaching. When it is leached with ferric sulphate, the dissolution takes place in two stages, which may be written:

$$Cu_2S + Fe_2(SO_4)_3 = CuSO_4 + 2FeSO_4 + CuS, \text{ and}$$

$$CuS + Fe_2(SO_4)_3 = CuSO_4 + 2FeSO_4 + S.$$

17 - Sullivan, J. D., Keck, W. E., and Oldright, G. L., Factors Governing the Entry of Solution into Ores during Leaching: Tech. Paper 441, Bureau of Mines, 1929, pp. 35-36.

The first reaction is much more rapid than the second. For particles of mineral 10 mesh or smaller, approximately 50 per cent of the total copper is converted into the soluble sulphate in 24 hours at 35° C., whereas approximately 20 days is required to dissolve the other half. Particles as large as 2 or 3 mesh dissolve at only a slightly slower rate than minerals crushed as fine as minus 150 plus 200 mesh. An oxidizing agent is necessary to dissolve chalcocite. Sulphuric acid in the absence of oxygen or water even in the presence of oxygen have practically no dissolving effect upon the mineral. Sulphuric acid in the presence of excess atmospheric oxygen attacks the mineral, but does so more slowly than ferric sulphate solutions. At 35° C. the rate of dissolution is practically the same in ferric sulphate and in ferric chloride. The rate of dissolution increases with increases in temperature. In 48 hours 50, 59, and 87 per cent of the copper was dissolved from minus 100 plus 200 mesh chalcocite at 23, 35, and 50° C., respectively.

Bornite.--Data on the rate of dissolution of various sizes of bornite, Cu_5FeS_4, in acidified ferric sulphate are given in Table 3.

Table 3.--Dissolution of various sizes of bornite in a solution containing 1 per cent of iron as ferric sulphate plus 0.5 per cent of sulphuric acid; temperature, 35°C.

Time	Copper extracted, per cent			
	-2+3 mesh	-10+28 mesh	-100+200 mesh	-200 mesh
1 hour	4	19	27	29
3 hours	7	26	28	30
8 hours	12	30	31	33
24 hours	21	38	38	41
48 hours	32	53	51	56
3 days	43	61	60	66
4 days	51	66	65	72
5 days	58	69	69	77
7 days	68	74	78	87
10 days	78	84	94	97
14 days	86	97	98	99
21 days	94	99	99	99
28 days	97	100	99	100

The rate of dissolution of bornite is markedly increased by increases in temperature. When minus 100 plus 200 mesh bornite was leached with acidified ferric sulphate 64 per cent of the copper was dissolved in 1 day at 50° C., in 4 days at 35° C., and in 14 days at 23° C. Eighty per cent of the copper was dissolved in 6 hours at boiling temperature. Bornite dissolves more rapidly in ferric chloride than in ferric sulphate. Sulphuric acid plus air attack bornite more slowly than ferric sulphate solutions.

Covellite.--Table 4 gives data on the rate of dissolution of various sizes of covellite in acidified ferric sulphate.

Table 4.—Dissolution of various sizes of covellite in a solution containing 1 per cent of iron as ferric sulphate plus 0.5 per cent of sulphuric acid

Time	Temperature	Copper extracted, per cent			
		-3+10 mesh	-10+28 mesh	-100+200 mesh	-200 mesh
1 day	35	4	6	8	13
5 days	35	6	10	16	40
11 days	35	8	11	21	49
24 days	35[1]	9	13	25	56
31 days	50	12	18	44	75
39 days	50	14	21	50	82
47 days	50	16	24	57	87

1 - Temperature maintained at 35° C. for 24 days and then increased to 50° C.

The rate of dissolution of covellite increased with increases in temperature. For a given sample, 81 per cent of the copper was extracted in 14 hours at 98° C., 81 per cent in 22 days at 50° C., and 41 per cent in 24 days at 35° C. The rate of dissolution was more rapid in ferric sulphate than in ferric chloride at 35° C., but the rates were virtually the same at 98° C. Covellite dissolved in sulphuric acid in the presence of excess atmospheric oxygen about half as rapidly as in ferric sulphate.

Chalcopyrite.—Chalcopyrite is ffequently found in leaching ores, but it is not appreciably attacked by common solvents at ordinary temperatures. Chalcopyrite is a closely textured mineral, and its rate of dissolution is greatly increased by fine grinding. Data on the rate of dissolution of chalcopyrite are given in Table 5.

Table 5.—Data on dissolution of chalcopyrite

Size, mesh	Temperature, °C.	Solvent	Copper dissolved in stated time, per cent
-100+200	35	0.5% of iron as $Fe_2(SO_4)_3$ plus 0.5% of H_2SO_4	1.8 in 45 days
-100+200	35	1.0% do. 0.5% do.	1.6 in 45 days
-350	35	2.0% do. 0.5% do.	28.5 in 25 days
-350	35	2.0% do. 0.5% do.	39.2 in 57 days
-100+200	50	1.0% do. 0.5% do.	4.6 in 8 days
-100+200	50	1.0% do. 0.5% do.	6.2 in 14 days
-350	50	1.0% do. 0.5% do.	32.2 in 8 days
-350	50	1.0% do. 0.5% do.	43.6 in 14 days
-350	35	1.0% of iron as $FeCl_3$	32.6 in 25 days
-350	35	1.0% of iron as $FeCl_3$	45.3 in 57 days

Figure 1.--Original minus 2 plus 3 mesh bornite before leaching

Figure 2.--Minus 2 plus 3 mesh bornite after leaching for 14 days at 50° C. with acidified ferric sulphate

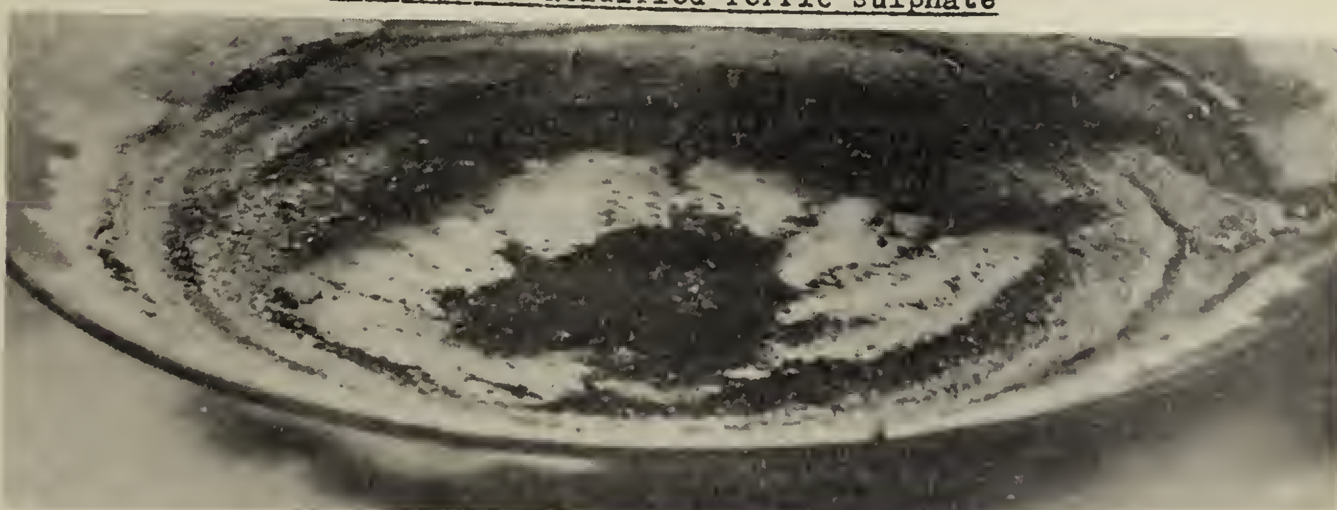

Figure 3.- Residue from leaching minus 2 plus 3 mesh bornite after treatment with carbon bisulphide

The foregoing data show that the rate of dissolution is faster when the mineral is more finely ground, that the rate increases with an increase in temperature, and that the rate of dissolution is more rapid in ferric chloride than in ferric sulphate.

In heap leaching an appreciable amount of the copper in chalcopyrite may be extracted owing to the long period of contact with solutions, and the possible rise in temperature of the ore during weathering.

When chalcocite is leached with ferric sulphate one-half of the copper dissolves rapidly leaving a residue approximating CuS which dissolves much slower; with bornite the copper is attacked preferentially to the iron; while with chalcopyrite the mineral dissolves as chalcopyrite; that is, at any stage in the dissolution(up to an extraction of approximately 45 per cent of the copper which has been the maximum studied), the ratio of Cu:Fe:S in the residue remains the same as in the original mineral. With covellite at any stage in the dissolution the residue has approximately the same ratio of Cu:S as the original mineral.

When large pieces of chalcocite, bornite, or covellite are leached with ferric sulphate the particles do not disintegrate and the residue become a powder. The particles, after leaching, nearly retain their original outline, and very little powder is formed. Apparently the sulphur left behind retains the original form of the mineral particles. When carbon bisulphide is added to the residue, the sulphur dissolves and the particles collapse. Figures 1, 2, and 3 demonstrate this phenomenon. Figure 1 shows minus 2 plus 3 mesh bornite before leaching; Figure 2, the residue after leaching for 14 days at 50° C. with acidified ferric sulphate; Figure 3, the residue after treatment with carbon bisulphide. In the 14-day period of leaching 91 per cent of the copper was dissolved.

REMOVAL OF SOLUBLE COPPER FROM LEACHED ORES

In heap leaching, because the solution is added to a given section of the heap and because after a certain quantity of solution has been added the leach solution is added to another section, several months may elapse between additions of solution to a given section. Heap leaching, therefore, results in a process of alternate wetting and drying. When the surface is dried, evaporation pulls solution to the surface where the dissolved salts crystallize as the solutions evaporate, and the next wetting operation removes part of these salts. At the same time that the salts are removed by washing some solution penetrates into the cavities, cleavage planes, and pores within the ore. If all of the moisture within the ore was removed during the drying cycle, all of the copper salts would be precipitated, partly at the surface and partly within the ore. When the ore is washed, the distance that the solution penetrates into the ore is a function of the time of washing.

The alternate wetting and drying tests carried out in the laboratory could not easily be duplicated in heap leaching. In the laboratory the surfaces of the ore may be completely dried, and it is possible to dry the entire sample.

In heap leaching, a complete drying would be practically impossible. As a heap may contain several million tons of ore complete drying, even of the surface, could hardly be expected. Experiments have shown that the copper can be brought to the surface even though the particles of ore are only partly dried.

With a 0.5-hour period of drying and a 0.5-hour period of washing and the cycle repeated until nearly all of the copper was extracted, 80 per cent of the total copper in solution was extracted from 3-inch pieces of ore in 6 hours, whereas 44 hours was required for 5 to 6 inch pieces. This shows the advantage of employing small pieces of ore.

In heap leaching, very short periods of alternate wetting and drying can not be maintained, but laboratory work has shown the advantage gained by keeping the cycles as short as possible. With 3-inch pieces of ore, an extraction of 80 per cent of the water-soluble copper was obtained in 6 hours with a 0.5-hour period of drying and a 0.5-hour period of washing, whereas 25 hours was required for a 6.0-hour period of drying and a 2.0-hour period of washing. Any advocacy of shorter cycles in alternate wetting and drying presupposes that the heaps are porous and well aerated.

The soluble copper can be removed by alternate wetting and drying in approximately 15 to 25 per cent of the time required to remove it by flood washing, provided the washing and drying periods are as close to each other as possible but long enough to permit a fairly thorough drying of the charge and soaking in of the leaching solution. As an example, it took approximately 130 hours to remove 90 per cent of the water-soluble copper by vat washing from the minus 1 plus $\frac{3}{4}$ inch size of a porphyry ore saturated with copper sulphate, whereas only $31\frac{1}{2}$ hours was required by alternate wetting and drying when the period of drying was 4.0 hours and the period of washing 0.5 hour.

A rapid movement of air past the surface of the ore promotes rapid drying. Anything that interferes with the circulation of air slows down the rate of extraction, thus demonstrating the necessity of having an open heap where free circulation of air is possible. Slime or other material that will coat the surface would also hinder drying. The rate of extraction is also increased by an increase in temperature.

When a rock is saturated with copper sulphate and then dried, the crystallized salt is disseminated throughout the entire rock, especially along the fractures and cleavage planes. A large part of the copper is brought to the surface in the first drying operation. Some of the copper is crystallized in the interior of the rock, especially along larger fractures and cleavage planes. Photographs demonstrating this phenomenon can be found elsewhere.[18]

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18 - Sullivan, J. D., and Sweet, A. J., Factors Governing Removal of Soluble Copper from Leached Ores: Tech. Paper 453, Bureau of Mines, 1929, p. 22 ff.

CONCLUSIONS

In leaching, the three factors (1) ingress of solutions, (2) dissolution of minerals, and (3) removal of the soluble copper are interdependent and work simultaneously. The speed of leaching can be no faster than the slowest step. Enough solution can not enter a rock at one time to dissolve the copper minerals completely unless the copper content of the ore is very low. A sample of ore weighing 2,000 grams may soak up about 40 cubic centimeters of solution. If the ore contains 1 per cent of copper, and the solution coming out of the pores contains 20 grams of copper per liter (assumed), 1,000 cubic centimeters of solution or approximately 25 renewals of the solution within the voids would be necessary. Frequent renewals of solution and much time is required to get sufficient solvent into contact with the minerals and to wash out the dissolved copper.

In heap leaching with oxidized ores the slowest step is probably the removal of the soluble salts. If the copper exists as sulphide (chalcopyrite excepted), about 20 days will be required for complete dissolution of the copper, granting contact of mineral and solvent solution during the entire period. In this instance, the slowest step is probably the chemical one or the dissolution of the copper sulphide, granting that conditions found in the laboratory hold in practice.

The rate of dissolution of the copper minerals is a chemical factor; and in heap leaching practice, for a given solvent, the rate can not be markedly changed. Although the rate of dissolution increases with increases in temperature, it is difficult to control temperature in a large heap. Actual temperature measurements within the heaps usually indicate a higher temperature than that of the outside air, which shows that the heat from chemical reactions is fairly well stored within the heap. It may therefore be possible to increase gradually the temperature within the heaps by adding warm solution. In the Southwest the solutions could be warmed by the sun in summer.

Waste smelter gases, if available, might be used with a heap leaching plant. If waste gases are allowed to enter heaps and displace the air present before water is added to a given section of the heap, the solutions should penetrate more rapidly into the interior of the rocks. Also, waste gases would add a reagent that would help to dissolve the copper minerals present.

From a practical standpoint, the method of speeding up the rate of extraction is to crush the ores finer. Crushing the ore particles would not only decrease the time required for saturation but it would also make the ore more amenable to leaching by open cleavage planes and crevices in the ore particles and by shortening the path required for the leaching solutions to come into contact with the mineral particles and for the dissolved copper to be brought to the surface. Crushing to a smaller size than 2 or 3 inches might produce too many fines, which would tend to counteract the advantages of crushing.

Another means of speeding up the extraction is by recirculating solutions over the heap. Frequently mine water containing neither acid nor iron salts is passed over the ore, but owing to the weathering of pyrite and other sulphides there is a considerable concentration of sulphuric acid and salts of bivalent and trivalent iron in the effluent solutions. As chalcocite is amenable to leaching by solutions of salts of trivalent iron, there is good reason for recirculating at least a part of the effluent solutions. To avoid "plugging" the leaching column with flocculent hydrated iron precipitates, the solutions recirculated should contain some free acid.

Heaps should be laid down on prepared footings to prevent the loss of solutions by seepage. This is especially true if pregnant liquors containing copper are recirculated.

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Scott Turner, Director

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TWENTY LIVE REASONS FOR FIRST-AID TRAINING IN CALIFORNIA

BY

Emory Smith

I. C. 6426
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DEPARTMENT OF COMMERCE - BUREAU OF MINES

TWENTY LIVE REASONS FOR FIRST-AID TRAINING IN CALIFORNIA[1]

By Emory Smith[2]

INTRODUCTION

The preface of the United States Bureau of Mines Manual of First Aid Instruction contains the following paragraphs:

> One of the most important inquiries that Congress has authorized the Bureau of Mines to conduct relates to health conditions among those engaged in the mineral industries. Investigations made early with a view to bettering these conditions demonstrated the need of prompt care of injured miners, as, under the difficulties inherent to mining and other mineral industries, wounds seemingly unimportant, if not treated promptly, may become infected and possibly cause permanent crippling or even death of the person. In the past many seriously injured persons have been treated by fellow workers ignorant of approved means of treatment and transportation and have been handled in such a manner as to cause suffering and to accentuate injuries to such a degree as to bring about permanent disablement. Accordingly, the need of general instruction in first aid became apparent.
>
> It must be recognized that the conditions in and about the mines, mills, smelters, quarries, oil fields, etc., usually present difficulties not found in large industrial centers where experienced surgeons are available and hospitals close at hand. Many mines are isolated and, when an accident occurs several miles underground, there may be a serious lapse of time before the injured person can be brought to the surface and taken to the hospital or given attention by a surgeon who may have to come a long distance. Under such conditions it has been frequently demonstrated that, if effective first aid is not given, the injured person may die or become permanently crippled...

First-aid instruction has been found to be an excellent aid[3] in preventing accidents. In addition to the immediate relief of injured persons, it has been found that most persons while in the act of practicing dressings for

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6426."

2 Foreman miner, safety division, U. S. Bureau of Mines, Berkeley, Calif.

3 Murray, A. L., Some Phases of Accident Prevention in Industry: Information Circular 6055, Bureau of Mines, 1927, 3 pp.

injuries are very likely to become interested in the way such injuries are received. From this point onward, under an intelligent instructor, first-aid training becomes accident-prevention work of a highly effective nature. This latter feature has an especial appeal to many companies that are conscientiously working to reduce their accident rates, and the popularity of the training has grown with the years to such an extent that the bureau is now unable to provide enough instructors to meet the demand; this fact has been largely instrumental in causing many companies to include the course in the regular program of their safety department, using their own instructors to do the training.

To arrive at the real value of first-aid training, when used either as an accident-prevention measure or in actually dressing injured persons, consideration should be given to the effects of accidents on the community as well as on the mine or plant. The result of an accident, industrial or otherwise, is not confined to the mere physical suffering from the injury, but because of the time lost from work there is also a loss of wages, or if the injury is serious the earning power may be permanently impaired or destroyed and a case of dependency is very likely to result. The loss of wages affects not only the worker and his immediate family, but the lowered buying power of the family will also have its effect on the fortunes of the business men of the community. Thus, while the families of the injured wage earners will be the chief sufferers, the cost of all accidents must eventually be shared by the community. Directly or indirectly, the community must support those unable to support themselves, and the direct cost of supporting the crippled inmate at some institution is probably no greater than the indirect cost of supporting the crippled street peddler.

It is customary, when a first-aid class is being planned or under way, to stress the value of the training as an accident-prevention measure, and when a comparison can be made of the frequency or severity rates before and after the campaign, the comparative figures in many instances bear out the assertion that universal first-aid training is one of the greatest accident prevention measures that can be recommended by the Bureau of Mines.

This widespread use of first-aid training as a means of achieving industrial safety has increased in recent years to such proportions that it tends to obscure the original purpose of the training, which was simply to prevent unnecessary suffering or loss of life pending arrival of the doctor. While it is true that an ounce of prevention is worth a pound of cure, once the accident has occurred the pound of cure becomes an urgent necessity which only the trained person can supply.

Moreover, it should be remembered that the knowledge of first-aid methods, gained at classes fostered by any agency, is equally applicable in industrial and nonindustrial accidents. The frequency and severity rates will show the effects of first-aid training on the accident record of the mine or plant, but such charts do not reflect the benefits of successful application of first-aid treatment at home or in the general walks of life. The accident-prevention

records of a company, therefore, do not show the true measure of service rendered to a community by a company which includes first-aid training in its safety work.

This has been borne out in several instances where men trained in first aid at classes sponsored by their employers have been able to save lives in nonindustrial accidents. Two such instances (cases 1 and 2 below) occurred following first-aid classes conducted during the spring of 1929 by the Bureau of Mines for employees of the Associated Oil Co. at Ventura, Calif.

CASES OF APPLIED FIRST AID

Case 1 - Injury while off duty:

In November, 1929, a group of oil workers at Ventura, Calif., were engaged in a friendly scuffle while off duty. During the mixup one of the men thrust his right arm through a window, severing the artery just below the elbow. Tolliver F. Miller, an employee of the Associated Oil Co., who had attended the first-aid classes of the previous spring, happened to be present. Miller successfully applied a tourniquet to the injured arm and removed the injured man to a hospital for treatment.

Case 2 - Injury in the home:

Christie Theis was another of the Associated Oil Co.'s employees who attended the classes. In January, 1930, the daughter of his next door neighbor fell upon a piece of glass and cut the artery over her left eye. Unsuccessful attempts were made to check the bleeding by using compresses. Theis, who was just leaving for work, was called and successfully applied digital pressure to the artery at the temple, stopping the flow of blood until a doctor could be summoned.

Case 3 - Automobile accident:

On September 27, 1930, an automobile accident occurred in which a woman was thrown through the windshield of the car, severing the artery in the right side of her throat. Fred Thieme, of the Pacific Portland Cement Co., Redwood City, Calif., was riding in a car immediately behind the wrecked automobile. Thieme managed to pinch the artery between his fingers, and when an ambulance could not reach the scene of the accident because of the extremely heavy traffic, he accompanied the woman while she was being transported to a hospital in an ordinary automobile, successfully holding the artery for half an hour.

Case 4 - Emergency in the home:

In July, 1929, Judson Conner attended the first-aid classes conducted by the Bureau of Mines at the Redwood City, Calif., plant of the Pacific Portland Cement Co. Conner began his attendance on Monday; on Tuesday he learned how to

remove foreign matter from the throat of choking persons; and on Wednesday a piece of phlegm slipped into the windpipe of his infant daughter, resulting in suffocation. Conner returned home at this critical time, and finding the child's face black, remembered his lesson of the previous day. He picked the baby up by the heels and slapped her smartly between the shoulders, successfully dislodging the phlegm.

Not always does the first-aid man have an opportunity to exercise his knowledge while the course is so fresh in his mind.

Case 5 - Electrocution:

In this case the first-aid man, who was also trained by the Bureau of Mines, found no practical use for the training for over 12 years, yet when an emergency arose he remembered his training and saved a life. At one of the diatomaceous earth quarries of the Celite Corporation, Lompoc, Calif., the insulation wore off a power line carrying 440 volts, and on March 15, 1929, the switchbox of a dragline scraper became charged so that the current grounded through the body of the dragline operator when he attempted to work the starting box lever. He was discovered, unconscious, not breathing, by Felice Pivato, a fellow worker who investigated when his signals were not answered. Pivato first broke the circuit by opening the line switch, and then placing the unconscious man on his face, administered the Schaefer method of artificial respiration for about 20 minutes until breathing was resumed, after which he sent the injured man to the hospital.

ELECTROCUTION ACCIDENTS FROM HIGH-TENSION LINES

The Southern California Edison Co., a public utility corporation with mining affiliations making its employees eligible for Bureau of Mines training, employs a very capable first-aid instructor who has trained several hundred men. The records of this company show that it has five men who hold the coveted Insull medal, which is awarded only to men who have successfully applied the Schaefer method of artificial respiration to men who have been electrocuted.

Case 6:

On November 15, 1923, an electrician came in contact with a conductor carrying 16,000 volts at the substation at Santa Paula, Calif. The substation operator, J. C. Gaertner, administered the Schaefer method of artificial respiration for 15 minutes, until breathing was resumed by the patient.

Case 7:

On March 30, 1925, a lineman came in contact with a conductor carrying 11,000 volts at the substation at Katella, Calif., and was rendered unconscious. The substation operator, J. G. Rhoday, administered the Schaefer method of artificial respiration for about 30 minutes. The patient recovered.

Case 8:

On February 5, 1927, a lineman who was working on a pole at Monrovia, Calif., came in contact with a line carrying 4,000 volts. The gang foreman, E. F. Fees, successfully removed the unconscious injured man from the pole and administered the Schaefer method of artificial respiration for about 20 minutes, after which the patient resumed breathing.

Case 9:

On January 12, 1928, a lineman at Compton, Calif., in removing the slack from a wire pulled it in contact with a line carrying 4,000 volts and was rendered unconscious. The senior lineman, Fred R. Jones, applied the Schaefer method of artificial respiration for about 10 minutes, and the patient recovered.

Case 10:

On November 5, 1929, an electrician employed by an oil company at Santa Fe Springs, Calif., came in contact with a conductor carrying 4,000 volts. E. W. Horsley, a service-gang foreman of the Southern California Edison Co., and his assistant, J. W. Ferguson, applied the Schaefer method of artificial respiration to the unconscious man until he recovered, about 20 minutes later.

The operator in case 7 was trained by the Bureau of Mines, while the operators in cases 3, 4, 5, and 6 as well as in cases 11 and 12, were trained by the company instructor. Cases 10 and 11 are also examples of the service rendered to a community by an industrial concern training its employees in first aid to the injured.

Case 11 - Drowning boy revived:

On August 17, 1928, as Pat Crowe, a "trouble" man employed by the Southern California Edison Co., was standing in a drug store, a man rushed in saying that a boy had been drowned in a swimming pool a few blocks away. Crowe and the druggist immediately drove to the pool, where they found the life guard and others who had worked on the boy, and who had given him up for dead. Crowe shook the boy to dislodge any water from the lungs, applied the Schaefer method of artificial respiration for only five minutes, and the boy resumed breathing.

Case 12 - Struck by a truck:

The Southern California Edison Co. instructor also found time to train members of the Huntington Park, Calif., fire department in the Bureau of Mines methods of first aid, and this training was instrumental in saving a life when a traffic accident occurred in front of the fire station.

On March 27, 1930, at Huntington Park, Calif., an autoist left his car and started across the street in front of the fire house. With his mind apparently on matters other than traffic, he walked into the side of a passing truck, one corner of which struck him in the head, inflicting a basic fracture of the skull and severing an artery at the temple. Members of the fire crew witnessed the accident, and Captain Charles Holcomb, assisted by Willard Lavenbarg, William Harris, and William Julian, firemen, applied digital pressure, successfully stopping the flow of blood, and otherwise cared for the injured man until arrival of the ambulance. The patient recovered.

Case 13 - Asphyxiation by gasoline vapors:

The Shell Oil Co. of California has for many years been interested in first-aid training, and in addition to encouraging its employees to attend classes conducted by the Bureau of Mines, this company has a number of very capable instructors who conduct classes each year. Men trained at these classes have successfully treated the victims of several accidents, both at work and away from it, where the life of the injured person depended upon prompt and efficient first-aid treatment.

On October 1, 1925, one of the men employed at the Watson Refinery of the Shell Oil Co. was overcome by casing-head gas while working in a tank car. He was rescued unconscious and not breathing, and Dan Sheldon, a stabilizer operator, performed the Schaefer method of artificial respiration for about 30 minutes, until breathing was resumed. In this case an oxygen inhaler was used with the artificial respiration.

Case 14 - Throat cut:

On October 25, 1927, an automobile accident occurred in Los Angeles, Calif., in which a man was thrown through the windshield of his car, severing the carotid artery in the right side of his throat. C. G. Bateman, an employee of the Shell Oil Co. at the Watson Refinery, who had been trained in first aid both by the Bureau of Mines and by the company instructor, was present and managed to grasp the end of the severed artery in his fingers, successfully pinching it down for about five minutes while the injured man was being rushed to the hospital in a police car. The patient recovered.

Case 15 - Asphyxiation by hydrogen sulphide:

On December 12, 1927, a workman at the Dominguez Refinery of the Shell Oil Co. was overcome by hydrogen sulphide gas while working in a valve box. The victim, who had discarded the gas mask provided for his safety, was rescued by nearby workers and removed to fresh air where the Schaefer method of artificial respiration was applied with ultimate success in an hour and 20 minutes by Fire Chief B. E. Bernius, Assistant Fire Chief C. H. Pierce, F. L. Maddy, and William Curry, all employees of the Shell Oil Co., who had been trained in first aid by the Bureau of Mines. Artificial respiration in this case was supplemented by an oxygen inhalor.

The Shell Oil Co. has expended considerable effort in teaching first aid to its field as well as refinery employees, and that this training has been the means of saving at least four lives is indicated in the following statement of results attained:

Case 16 - Electrocution by a live wire on the street:

William Avery, well puller at Long Beach, saved the life of a lineman repairing an electric wire in front of his house. The lineman came in contact with a live wire and fell from the pole to the ground. He was unconscious from the shock. Avery ran from the house and gave the injured man artificial respiration which saved his life.

Cases 17 and 18 - Arteries severed in automobile accidents:

Howard Wineland, garage mechanic, Santa Fe Springs, two days after he received first-aid instructions applied a tourniquet to a child who had received a severed artery in an automobile accident. He also applied a tourniquet on another man injured in an automobile accident. In both cases the doctor treating them stated that the application of the tourniquet saved the lives of the patients.

Case 19 - Drowning Baby revived:

In another case, Pete Hammer, pipe line foreman, Santa Fe Springs and Brea, three days after receiving first-aid training saved his child's life in the following manner: The child was in the bath tub and Hammer was reading the newspaper in another room. He did not hear any splashing of water or any sound whatever where the baby was and he investigated the matter. He found the baby was under the water and its body purple. He remembered his instructions in resuscitating drowning persons, and before the doctor arrived, Mr. Hammer had succeeded in saving the life of the baby, who to all appearances had ceased breathing.

Case 20 - Asphyxiation while cleaning oil tank:

The Richfield Oil Co., one of the major oil companies operating in California, employs very competent first-aid instructors who make a determined effort to reach every employee with the training. The operator in this case, who was trained by the safety engineer of the Richfield Oil Co., received the President's Medal, awarded by the National Safety Council for meritorious service in the saving of life.

On September 26, 1929, an employee of an oil company was engaged in cleaning a tank at Davidson City, Calif. The man was provided with a hose mask but in some unknown manner the hose became disconnected from the face piece. He immediately signaled his attendant on top of the tank but the attendant was unable to pull him out alone. The attendant's cries, however, attracted the attention of Fred E. Keithley, an employee of the Richfield Oil Co., who rushed

over and assisted in removing the man from the tank. The top of the tank was covered with 4 inches of water, but as the man was not breathing Keithley did not waste time moving him elsewhere, but laid him on a ladder which was lying on the tank top and administered the Schaefer method of artificial respiration. After 15 minutes the patient showed signs of life and after 25 minutes he could breathe without assistance. The patient was not an employee of the Richfield Oil Co.

CONCLUSION

The foregoing list of emergencies in which first aid has been administered is far from complete, as no effort has been made to uncover any cases other than those in which the Bureau of Mines training has directly entered. Many agencies are teaching methods of first-aid treatment, bringing standard methods of instruction within reach of practically everyone interested. One point generally overlooked, however, is that companies interested in accident prevention and in proper care of injured employees are by their efforts creating a group of trained first-aid men who are a distinct asset to the community.

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April, 1931

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

SAFETY CONSCIOUSNESS

BY

F. S. Crawford

I. C. 6427
April, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SAFETY CONSCIOUSNESS[1]

By F. S. Crawford[2]

Safety devices and guards are often looked upon as the last word in safety by the men around the shop or large industrial plant. However, thorough knowledge of the details of their jobs and of the dangers attending them are even more important.

It should be unnecessary for all men to go through the experience of the first man to meet an unknown danger, but that appears to be about the only way by which many learn. Practically every danger connected with the job around a mine, blast furnace, steel mill, foundry, coke oven, cement mill, or other operation, was encountered for the first time by someone, and if he was fairly fortunate he either got hurt or just missed being injured and lived to tell his "buddies" how it happened; he spread the knowledge of that particular danger. Sometimes, but not often, a man can foresee a danger. If he could always be on guard and remember everything he had ever learned and also know everything that had happened to the other fellow, and then let that knowledge keep him aware of the dangers in his own immediate "job," so that he would work the safe way for both himself and the other fellow, he would have what safety engineers call "safety consciousness."

Being safety conscious is a fine attribute, as it really means knowing all about one's job. A man who is really safety conscious is one of the best and most efficient workmen in the whole plant. The boss wants men of that kind--men who want to learn and who try to learn every little detail about their job--where there is danger of spoiling the heat or getting burned by hot metal in a steel plant or of loading rock instead of coal or ore in a mine. Such men know how to do whatever job they are on easily and with the least amount of hard work and at the same time get it done as quickly as possible with safety. A man who is safety conscious has to nearly all intents and purposes developed eyes in the back of his head, if he is in a big mill where there if something going on all around him. These eyes are not real eyes, but they are what might be termed a sixth sense which he has developed by his experience in the mill as well as in other dangerous work which he may have done before in other plants.

There may be occasions when a man's safety consciousness will leave him for just a short time; this is likely to happen when he is thinking about a quarrel he had with his wife which has not been cleared up happily, or perhaps when he has bought too many things on installment and with sickness at home

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular No. 6427."

2 - District engineer, United States Bureau of Mines, Duluth, Minn.

can not meet all his bills. It may happen when the day is hot and sultry, or when it is bitterly cold. When the weather is hot perhaps a man will not have as clear a brain as in more moderate weather, and he should be more careful. When it is cold, he is apt to be numbed or cramped, or his movements may be hampered by heavy clothing; he may not be in the best physical condition, and he may not have his mind on his work. ....

To guard against temporary lapses of memory and inattention, which may result from poor physical condition or worry, the proper thing to do in any operation is to guard everything where guarding is possible. Install safety devices, make systematic and periodic inspections of the condition of machinery, and replace worn parts long before they may be expected to fail from being worn out and thus cause the loss of life or limb and perhaps damage to other property.

In spite of all the guarding we may do occasionally some man has an accident and is killed or badly injured; something may fall accidentally, and if a man is in the way he will get hurt. If the man is not in the way the accident may happen just the same, and while nobody will get hurt there is likely to be some property damage. Strive to look ahead for the possibility of an accident and if possible prevent it; then nobody gets hurt by being in the way because nothing happens that is not foreseen.

There are really very few true accidents because an accident is an occurrence out of the ordinary, which could not be foreseen. If we take into account the possibilities of accidents, we will stop nearly all possible chance of accidents occurring and be prepared to handle those other hazards that can not be eliminated. After a man has had enough experience and ability to qualify him to perform his assigned duties he is supposed to be able to foresee what will happen in the ordinary run of his work and be prepared to take certain precautions against things that would happen if he did not take the proper precautions. Although guards and safety devices have their proper place they are useless unless a man employs them right. Guards and safety devices are essential. They should be kept in place and in operation at all times to take care of the sudden failures man would not have time to handle because they happened so quickly, but the safety device is useless unless the man responsible for its operation takes the necessary precautions to see that it is in working order at all times.

The foreman in any plant bears the immediate responsibility for the safety of his men and for efficient work. He should know the details of all the jobs under him and be able to instruct each of his men in the dangers and difficulties in his individual job. To impress upon a man some of the dangers in his job it is sometimes well to give him a practical demonstration, if that is possible without ruining machinery or other property. Although that demonstration does not replace a guard, it is more effective with the workman than any mechanical guard because he has been given a mental picture of the danger and should know what dangers to avoid. After all, the things which we know best are those which we have actually experienced; if the experience can be gained under the safe conditions of a demonstration a man will remember the demonstration better than something about which he has merely been warned.

We ought to stop occasionally when we are thinking about safety in our industrial operations to consider what has happened in past history. If we had enough records we could trace our ancestors for countless years. Nobody really knows how the first man started, but the men who dig up rocks with the skeletons of animals in them, which have been turned to stone, have put all their findings together and worked up a science called "geology." These men tell us the world is many millions of years old and that men appeared in the world much later than the creation of the world itself. The first men lived in jungles and had to fight fierce wild animals with their bare hands, but they had brains and intelligence enough to learn something from every hair-breadth escape. The men who didn't have the intelligence to learn from experience didn't live long enough to be fathers, so everyone now living is descended from jungle men who had enough safety consciousness to avoid wild animals.

Today we should take a lesson from the lives of those remote ancestors of ours and do our part in taming the "jungle animals" of our daily civilized life. The wild beasts of today are all about us, but if we keep alert and familiar with the details of our job we can keep them so far in the background that we will hardly know they exist. The fierce beasts about our plants to-day are the dangers surrounding us, ready to come into action when we make a mistake. Another way of looking at this is that when we break a law of nature something is bound to happen. Learn the laws of nature, and you can avoid accidents; the fact that a casting will fall to the floor unless a strong chain is holding it up is one of the laws of nature as is the fact that loose overhead material in a mine will fall unless it is properly timbered. If a man is under the casting or the loose overhead material when it falls he will get killed or injured.

No one who has learned by experience to be careful, realizes that the laws of nature are always in force, and knows what to expect goes around the world in fear and trembling thinking he is going to be hit any minute. A man who learns to be safety conscious keeps his mind open to new ideas, finds the best and safest way of doing the jobs which he has to do every day, and then starts a good habit by doing the job the right way every day. If he finds by unfortunate experience or from conversation with the superintendent, foreman, safety engineer, or some other workman that he is doing some parts of his job unsafely, if he is of the right type he changes to the safe way at once and makes that good habit take the place of the old bad habit.

It should not be necessary for every man to go through the dangers others have experienced before him to become a safe worker; if he is to be a safe worker he must go through these dangers mentally, after learning what has happened to others and may happen to him. Therefore, after this he does his work with his eyes open and senses alert, ready to recognize the danger someone else has pointed out to him in advance. In that way he will be using his mind and will get the experience he needs without getting hurt at the same time.

What every man needs and should try to get of his own accord is education; he should aim to know all he possibly can about what has happened in his own occupation in the past and keep learning new things about his job from time to time.

From the record of what has happened in the past we learn what may happen in the future. By studying accident records and trying to find the real cause of accidents we will get some value out of them and some education without going through the actual danger. We get the thought into our minds that "Here's something for which I must watch out." If we let our minds picture the possibilities of injury from the new thing we have heard we will at times/at least get it to stick in the back of our brain, where it is ready to jump up like a guard whenever we encounter that particular danger.

To get reports on injuries caused by accidents is not enough; every injury, besides being given immediate first-aid treatment, should be followed by an accident report. In addition, every man should be encouraged to make reports on slips, falls, material falling, and an endless number of other things which fortunately happened without injuring anyone but are unexpected. In that way, an effective safety department can build up a plant experience which can be passed on to the whole plant in the form of education as well as guards and alterations in methods of working which will make the chances of injury less from the accidents reported.

Nearly all accidents can be avoided if people will take the trouble to think beforehand. The brain is the best safeguard there is in the world if it is properly trained. If all workers would take the trouble to learn as much as possible about what they do and about their surroundings and keep on learning, instead of one safety inspector in a plant every man in it would be an assistant safety inspector and could pass on his ideas to the safety inspector, safety engineer, or superintendent. Two heads and two pairs of eyes are always better than one, and in a large mine or industrial plant a hundred brains and a hundred pairs of eyes can always have more thoughts and see more things than one. The worker should not leave it all up to the safety man or foreman. If some dangerous practice is seen warning should be passed along to the gang and the rest of the plant through the boss. Perhaps this will be the means of keeping a child happy because some dangerous condition has been reported which would have killed or badly injured his "daddy."

One of the worst results of the accidents in our industrial plants, mines, and factories, is that young mothers are compelled to neglect their children while they work to get enough to feed them. To see a young unmarried man suddenly struck down is regrettable in the extreme; he has his loved ones too, but he at least has not the responsibility of a family.

No matter what their race or the degree of their intelligence, every man and woman cares for someone and wants to keep that person with them. This is true of all the bosses and officials of mines and industrial plants, as well as of the workers. The big trouble with some of the heads of factories, mills, and mines is that they are so busy trying to make both ends meet that they feel forced to make safety work wait until they have the mine or plant on a better paying basis. They are just as sorry as anyone else that John Smith got killed because the hoist rope broke when he was on the cage or because some machine too light for the job broke down. The trouble with such managers is that they

haven't yet realized it is possible to run plants safely as well as economically and make more money than they could by allowing unsafe conditions and methods of working to continue; in other words they have not yet found safety consciousness. There are far too many managers who have put little or no time or thought into safety work and are probably deep in a rut of complacency out of which they will not budge; they are jogging along content to pay the compensation and insurance the State requires and believe that having paid the medical and compensation charges they have done their full duty. They believe that an accidental injury is an "act of God." It takes a hard jolt to make many men of this type realize that they ought to be preventing men from being killed and injured instead of merely paying the bill and that to do a good job of safety work the efficient manager must give freely of personal time and effort. Once a manager is aroused to the need of expending personal effort as well as instructions on safety work, and experiences the happiness that comes with knowing that he is keeping the families of his employees happy, the safety work goes forward. In some of our largest operations men are not killed, and work for years without losing a day's time from injury. Usually under these conditions there is an appreciable dollars and cents saving to the plant.

Six years ago one of the largest limestone companies in the world was working complacently along having about 50 comparatively serious accidents and some deaths every year. About that time the president of the company was aroused to his responsibility for the safety of the men working under him by a visit from the safety director of one of the factories to which he sold limestone, where they had an enviable record for safe working. After this meeting he started the safety work in his own quarry and other operations connected with the plant. A safety director was appointed, central and departmental safety committee meetings were started, and results began to be apparent. The output increased every year, the accidents decreased every year, and the lost time from delays in operation was decreased. In 1929 this company had no lost time from accidents in its quarry and screening department, which made up practically all of the operation. In all operations there were only two accidents in 1929. Up to September, 1930, there was only one in all departments for the year. The biggest factor in attaining a practically perfect safety record unquestionably was the active aggressive personal interest of the president of the company.

The head officials of any company must be squarely behind the safety movement or it generally gets nowhere. Next in importance to the active leadership of the president at this limestone quarry was the alert interest of every man in the operation, obtained through his freedom to express his ideas when serving on his departmental safety committee and the freedom of the department managers to express ideas in the central safety committee. Action upon the ideas found to be sound followed the committee meetings.

Every mine or plant should have a central safety committee headed by the highest official and composed of the heads of the main divisions in the plant. This committee should meet monthly or twice a month and the "highest official" should by all means be present and participate actively in all or

nearly all of these meetings. Each minor division should have its own safety committee which meets weekly to go over its own portion of the plant. On their trip through the plant, members of the safety committee should talk to all or nearly all of the men and get their suggestions for increasing safety. They should also caution and correct men who are working unsafely. They should then hold a short meeting, recording their observations for the central safety committee. Anything which can be corrected immediately without waiting for the indorsement of the central safety committee should be handled without delay. The central safety committee acts largely as a clearing house for ideas originating in one department to be passed on to the others.

By periodical analysis of conditions and a willingness to face the facts and to apply the remedy combined with the willing cooperation of all of those in the mine or plant from the highest officials to the lowliest worker, lives are saved and families kept happy. Every mine or industrial worker should do his part when called on to serve on a safety committee and should also pass on his ideas to the committee when not serving on it. Every official should do his utmost to see that all conditions in and around the mine and plant are as nearly accident proof as feasible and should not stint personal effort to bring about this happy, and in the end, paying, situation. When this spirit pervades the organization it can be said to have acquired safety consciousness. If our jungle fathers were able to be safety conscious, surely their descendants should be able to do as well.

I. C. 6428 April, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

THE PARAMOUNT ISSUE

BY

W. D. RYAN

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

THE PARAMOUNT ISSUE[1/]

By W. D. Ryan[2/]

The American public has at all times been prone to discuss important questions that call for consideration and adjustment. Debatable subjects upon which sides may be taken always receive unusual prominence, but other questions equally important may be neglected. The real "paramount issue," calling for remedial action, is the ever-increasing number of deaths caused by preventable accidents.

This matter demands the attention of the entire American people; the importance of accident prevention can not be overemphasized, and in many phases of American life, especially among the mining and allied industries, far too little real attention has been given the importance of conserving life and limb by accident prevention.

ACCIDENTS COMPARED TO WAR AS A CAUSE OF DEATH

Harry T. Hoffman, Wyoming labor commissioner, writes the following:

> It is one of the peculiar complexes of this rapidly moving age that we all stand aghast at the horrors of war and view with apparent indifference the greater havoc of industrial peace. Did you ever stop to think that approximately twice as many people were killed by accidents in the United States in 1929, as there were American soldiers killed in the entire period of the World War?
>
> **********
>
> Every employer and every worker should be thoroughly aroused over the tremendous toll levied against industrial forces by accidents which are largely preventable and develop a faith in safety efforts that will fire them with a zeal as pronounced as that of their religion or their patriotism.
>
> Let us remember safety-first to the end that the workman shall live to enjoy the fruits of his labor; that his mother shall have the comfort of his arm in her age; that his wife shall not be untimely a widow; that his children shall have a father; that cripples and helpless wrecks who were strong men shall no longer be a by-product of industry.

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6428."

2/ U. S. Bureau of Mines mine safety commissioner, Kansas City, Mo.

To accomplish this it is necessary that every possible protection be provided for those who toil, and that every one should exercise the greatest care. It should be the workers' constant aim.

ACCIDENTS COMPARED TO DISEASE AS A CAUSE OF DEATH

Accidents are sixth among causes of deaths in the United States, according to figures compiled by the National Safety Council from records of the United States Census Bureau. Other principal death causes are heart disease, tuberculosis, cancer, cerebral hemorrhage, and nephritis.

Among all men, accidents are third as a cause of death, being exceeded only by heart disease and tuberculosis. Among men from 20 to 45 years of age accidents are the second cause of death, being exceeded only by tuberculosis.

For the ages 15 to 19 among men and for 5 to 14 in both sexes accidents cause more deaths than any one disease. For ages under 5, accidental deaths are second only to the so-called diseases of infancy.

It is only after the ages of 45 to 55 that heart disease, cancer, and other diseases become important and cause higher death rates than accidents. Of various classes of accidents, automobile accidents predominate as a cause of death for ages from 3 to 70.

PEOPLE GENERALLY NOT CONSCIOUS OF IMPORTANCE OF ACCIDENT PREVENTION

Accident prevention is not, however, prominent in the public mind because it does not have the dramatic appeal of a purely political question. We receive our impressions of it from single instances and isolated occurrences; and only the student or the searcher for truth knows of its human horror or its economic importance as a whole. It is difficult to focus attention upon it. The attitude of the average individual toward the problem of accident prevention is somewhat similar to his attitude toward death; he is willing to admit, at any time, if he is pressed to do so, that death may be very near, but he thinks and acts always as if it were very remote. Thus any particular individual may not have suffered directly or indirectly from any particular accident. His acquaintance with the subject as a whole may come from reading an account in his daily paper, and he probably views the matter in an impersonal light. He is utterly unconscious of his own nearness to it. He probably does not realize that the cumulative cost of all the accidents happening in the country every year is tremendous and that every individual, although he may never have witnessed an accident, pays his share of the overwhelming price.

ENORMITY OF HUMAN LOSS

If a catastrophe causing the loss of one thousand lives happened on Monday morning in any part of the country the news would be flashed by telegraph and radio throughout the world. If another such accident, involving essentially the same loss of life, happened on Thursday of the same week, the country would be shocked. If another happened the first of the next week and still another the latter part of that week, public attention would be centered upon the unusual happenings and there would be little room in the public press or the public mind for anything else. If these disasters persisted with comparative regularity, the country would soon set about doing something; public attention would crystallize into action, relief would be organized, and effective measures of prevention would be projected.

In effect that is essentially what is happening; the total of all the fatal accidents that happen every week equals approximately two catastrophes with 1,000 persons killed in each. And this happens week after week. There were almost 100,000 fatal accidents in 1929, and apparently even that gruesome record will be exceeded in 1930.

The difficulty of reducing accidents lies in bringing a true conception of the meaning of this enormous human loss to every individual. With the taking of a human life, suddenly and without warning, there is associated the idea of horror, and usually in a fatal accident there are gruesome details that add to this repulsiveness. Any person who has witnessed an accident carries away an impression of this horror. But very few people take the time or trouble to try to form an adequate conception of what this horror would be when multiplied one hundred thousand times. It is something that beggars description and defies comparison.

ENORMITY OF ECONOMIC LOSS

So much for the human loss. What of the economic loss involved? Any attempt to measure this loss will prove that it is almost immeasurable and any results that may be obtained in the way of figures are of relative importance only, in that they may be used for purposes of comparison and possibly to suggest what the actual loss may be.

Economists assert that a definite value may be assigned to each human life, according to its remaining earning capacity and its ability to advance the welfare of the race.

Statisticians of the Metropolitan Life Insurance Co. have prepared the following table, assuming probable future earning of $2,500.00 per year:

Age	Life value
18	$28,654
21	30,818
30	31,038
40	25,795
50	17,510
60	8,400
70	562

It may be contended that these values are too high to assign to lives removed by fatal accidents. On the other hand, it is felt that the amount ordinarily used in computing industrial losses - $6,000 - is too low, especially when consideration is given to the record of verdicts by juries that have been as high as $100,000 on single lives. However, even if this amount, which seems pitiably small, is used the life values destroyed by accident every year amount to $6,000,000,000.

Many of the people killed by accident are not wage earners, but aged people, married women, and young children.

We may eliminate the first class entirely because the actual number of cases is so small as not to affect the result materially.

As to married women, even if we do not attribute any economic value to the status of housekeeper we can not deny value to the potentiality of motherhood. Moreover, irrespective of the possibility of future earnings of minor children there is always the possibility of their great future service to humanity.

Who, for instance, could have foretold the contribution of Nancy Hanks (Mrs. Thomas Lincoln) to the welfare of the country? Who can describe what the condition of our country to-day would be if Abraham Lincoln had not lived to maturity? What would be our industrial status if Thomas A. Edison had been the victim of a fatal accident in his youth? Little is known of the laws of heredity. Genius may appear among the rich and the poor alike and leadership comes both from the mansion and the cottage. But it is self-evident that a large number of American women and children can not be destroyed indiscriminately without suffering great loss of future leadership. In fact a single young life can not be destroyed without destroying the possibility of another Lincoln or Edison. This is one realm in which the laws of average and the rules of probability fail.

The value of the human asset generally is well illustrated by a statement made by Charles M. Schwab. When asked what he would do in case his steel properties were destroyed by fire, he replied:

> "I would not even figure those as a loss as ~~they~~ could all be replaced in time, but if some catastrophe should destroy at one stroke the personnel of our organization, I would consider myself a ruined man."

LOSS FROM NONFATAL ACCIDENTS

Added to this burden of fatal accidents is that caused by nonfatal or disabling accidents. It is estimated that there are 10,000,000 such accidents in the country every year and that 240,000,000 days are lost as a result. The actual loss in wages may only be estimated, but if the cost of medical attention and of other direct charges is included, the estimate must reach $1,000,000,000.

Therefore, there is a yearly loss of about $7,000,000,000 directly attributable to accidents. Is this the entire story? By no means.

Assume that among all the people killed by accidents each year only one half are wage earners. Assume further that each such wage earner is survived by two dependents and that the period of dependency in each case continues for five years. It will be admitted that each one of these three assumptions is conservative. That would give at the end of five years 500,000 persons dependent as the result of fatal accidents, and if experience did not improve this total would not decline. In addition, 240,000,000 working days lost through nonfatal accidents means the same as an idleness of 800,000 persons for 300 working days. That gives in all at least 1,300,000 people constantly dependent as the result of accidents. Who supports these dependent people?

It makes little difference whether or not some of these people may be apparently independent by reason of insurance or other savings, and it makes little difference whether contribution is made to indigent persons by the payment of insurance premiums or donations to charitable enterprises; this fact remains, that when the producers who normally, according to the usual expectancies of life, would support these people, are withdrawn by accident the burden of actually supporting them shifts to the producers who remain.

An attempt will not be made to measure what this "cost of support" may be, but it will be sufficient to call attention to the cost of $7,000,000,000 per year. What is the significance of this figure?

It is equal to approximately 7 per cent of the annual income of all the people of the country. A saving of less than 50 per cent of this amount each year would pay all the expenses of the national government or would supply the money necessary to solve many of the national relief problems.

Is it possible to prevent a considerable portion of this tremendous human and economic loss?

Experience gives an unmistakable answer in the affirmative.

ACCIDENTS CAN BE PREVENTED

Reference has been made to the ever-increasing death list caused by preventable accidents. During the past few years there has been an increase when all causes are included. At the same time we find a decrease in industrial accidents, the greatest increase coming from automobile traffic.

It is true that 75 per cent of all industrial accidents are preventable. This assertion is corroborated by the following records showing accident reduction on railroads of the United States.

Year	Employees		Passengers	
	Killed	Injured	Killed	Injured
1916	2,513	49,121	246	7,152
1917	2,781	52,780	301	7,582
1918	2,928	47,556	471	7,316
1919	1,759	36,601	273	7,456
1920	2,198	47,234	229	7,591
1921	1,137	28,747	205	5,584
1922	1,298	32,434	200	6,153
1923	1,645	39,734	138	5,847
1924	1,246	32,401	149	5,354
1925	1,299	32,484	171	4,952
1926	1,371	34,202	152	4,461
1927	1,238	28,157	88	3,893
1928	1,039	23,779	91	3,463

This decrease has been effected during a period when there has been a very large increase in traffic, so that the actual improvement is much greater than the above table indicates.

The National Cement Association has reduced accidents during the past six years about 75 per cent. Although exact figures are not available, the steel industry shows a material decrease. The coal industry suffered one fatality for every 313,000 tons produced during the year 1929; this in face of the fact that a number of coal companies produced from one to seven million tons without a fatality. There is no question that the coal-mining industry lags to a distressing extent in this matter of prevention of accidents in industry.

It is a proved fact that when effective methods of accident prevention are applied material results are practically certain of achievement.

When we consider automobile accidents from which the recent increase in total has come, we find this truth again demonstrated. Most automobile accidents are caused by private pleasure seekers as well as by employed drivers. The person who can be reached by education, supervision, and discipline causes relatively few accidents; the person who is not made conscious of responsibility at the steering wheel causes many.

In this connection attention is directed to the two outstanding organizations in the United States centering their efforts on safety and accident prevention. These organizations are the National Safety Council and the United States Bureau of Mines; the former covering all industries, including automobile traffic and the latter, mining and kindred industries.

People generally must be taught the significance of this problem of accident prevention. Too much publicity can not be given to make people conscious of the tremendous burdens assumed because of accidents, nor can too much instruction be given in the ways by which accidents can be prevented.

SUMMARY

The importance of accident prevention may be shown by comparing accidents with war and disease as a cause of death and disability and by consideration of the human and economic loss involved. That prevention is possible is proved by results obtained wherever effective measures are undertaken. Why can not the gains made in certain industries be extended to all industries and to the public generally?

Accident prevention overshadows all other issues that are brought to public view from time to time. Especially is this true when there is considered the human wasteage, the intense sorrow, and the suffering incident to this trail of misery, to say nothing of the enormous economic loss that accrues to the nation in all these varied aspects.

The fatal and nonfatal accident records of our country are overshadowing, and are appalling in consequences to the American people; these records should prompt the employers and workers alike to redouble their efforts to enlist the Federal and all State governments to grapple with this national problem in some tangible and effective way; it should be done in such manner that public opinion may be aroused with the result that safety consciousness will be instilled into the minds of every one in such a way as to be impelling and convincing.

An intensified movement of this kind would impress upon individuals in all walks of life the obligation of doing their utmost to eliminate this outstanding and unnecessary sacrifice of human life, as well as the economic consequences resulting therefrom.

Unquestionably if accidents are to be reduced to the lowest possible minimum, the spirit of "chance," which permeates the American people, must be eradicated. As statistics will show, much of our large toll of fatal and nonfatal accidents results from taking chances.

If the Federal and State governments undertake to handle this problem, as they should, with the help of enlightened public opinion, a long stride for human and economic progress will have been taken; and the reward will be bountiful, for the people of the nation will be greatly enriched by the human and economic results accruing from this endeavor, in making accident prevention the "paramount issue."

I. C. 6429 FEBRUARY, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

METHOD AND COST OF RECOVERING QUICKSILVER FROM LOW-GRADE ORE AT THE REDUCTION PLANT OF THE SULPHUR BANK SYNDICATE, CLEARLAKE, CALIF.

BY

WORTHEN BRADLEY

I.C.6429,
February, 1931.

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METHOD AND COST OF RECOVERING QUICKSILVER FROM LOW-GRADE ORE AT THE REDUCTION PLANT OF THE SULPHUR BANK SYNDICATE, CLEARLAKE, CALIF.[1]

By Worthen Bradley[2]

INTRODUCTION

This paper describing the metallurgical practice and costs at the reduction plant of the Sulphur Bank Syndicate, Clearlake, Calif., is one of a series of papers on the treatment of ores being prepared by the Bureau of Mines.

The reduction plant and mine of the Sulphur Bank Syndicate are on the eastern shore of Clear Lake, in Lake County, Calif., 4 miles from the town of Clearlake and 125 miles north of San Francisco.

About 30 men are employed at the mine and reduction plant. The average daily production of quicksilver for the year 1929 was 4.19 flasks, each flask containing 76 pounds, 1 ounce. The quicksilver is delivered by automobile truck, railroad, and boat to San Francisco for marketing.

The ore occurs as a surface deposit, covering about 100 acres to an average depth of 5 feet. Mining is done by the open-cut method with two 1 cubic yard shovels each operated by a 75-hp. Diesel engine. The ore is loaded by the shovels into automobile trucks, each truck having a capacity of 8 cubic yards. The trucks haul the ore about one-fourth of a mile to a bin at the lower terminal of an inclined tramway to the plant. The ore is dumped directly into the bin through an inclined grizzly with 9-inch openings; the grizzly is made of 70-pound rails. The oversize boulders slide over the grizzly onto a waste pile which is removed from time to time by the power shovels and trucks. From this bin the ore is loaded into 2-ton capacity skips and hoisted to the bin at the upper end of the plant.

The plant site is on the side of a hill about 1,000 feet from the shore of Clear Lake and 150 feet above it. The lake is 1,350 feet above sea level.

The capacity of the screening and sorting plant is 300 tons per day; the capacity of the furnace plant is 43 tons per day.

Water for the reduction plant is pumped from the lake to a height of 300 feet by a two-stage centrifugal pump direct connected to a 50-hp., 1,800-r.p.m. motor. The pipe line is 6 inches in diameter and discharges into two steel storage tanks each 26 feet in

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used. "Reprinted from U. S. Bureau of Mines Information Circular 6429."

2 - One of the consulting engineers, U. S. Bureau of Mines and manager, Sulphur Bank Syndicate.

diameter and 16 feet high and having a combined capacity of 127,000 gallons. About 75,000 gallons of water per day are required for use in the reduction plant.

Power is purchased from the Pacific Gas and Electric Co. It is transmitted to the plant at 11,000 volts and stepped down to 440 volts for all mine and plant motors.

ACKNOWLEDGMENTS

The author acknowledges the assistance of William O. Vanderburg, associate mining engineer, U. S. Bureau of Mines, and of W. H. Enderton, A. F. Wolbert, P. W. Cox, Jas. Storey, and C. W. Hart, of the Sulphur Bank staff, in the preparation of this paper.

ORE TREATED

The ore deposit consists of grains of cinnabar finely disseminated throughout a gangue composed principally of altered basalt with subordinate amounts of elemental sulphur and opal. The altered basalt is soft and porous and averages about 30 per cent moisture. Large bowlders of partly altered basalt which are barren of value occur in the deposit. The distribution of the cinnabar through the basalt is not uniform. Samples taken over the entire area covered by the ore deposit range from a trace to 10 pounds of quicksilver per ton, the average being about 0.1 per cent or 2 pounds per ton. Only that material which, after passing through a screen with 1-inch holes, will assay 6 pounds of mercury per ton or better is considered furnace ore. The cinnabar occurs mostly in the fines, as is shown by the following screen analysis of low-grade ore:

Screen analysis of low-grade ore

Screen size	Weight, per cent	Assay, pounds mercury per ton
On 2-inch, square mesh	30.5	1.75
Through 2-inch on 3/4-inch, square mesh	44.5	2.20
Through 3/4-inch on 1/4-inch, square mesh	11.8	4.25
Through 1/4-inch, square mesh	13.2	11.50
Composite	100.0	3.53

The free sulphur content in the ore varies from 1 to 15 per cent with an average of about 2 per cent. The ore is highly acid, and the action of the acid on the gangue results in a large amount of slime.

BRIEF HISTORY OF FURNACE OPERATIONS

The Sulphur Bank ore deposit was originally worked for sulphur as far back as the sixties, but difficulty was encountered in refining the product due to the presence of cinnabar. The deposit was first mined for the recovery of quicksilver from 1873 to 1883. Subsequent periods of operation were 1887 to 1897; 1899 to 1905; 1917 to 1918; and 1927 to the present time. Up to 1927, when the Sulphur Bank Syndicate acquired control of the property, the mine was credited with a quicksilver production of 92,400 flasks. In spite of the large production, considerable difficulty was experienced in furnace practice. The high free-sulphur content of the ore proved a barrier to complete separation of the quicksilver, as

part of the sulphur and mercury would recombine in the condensers, forming synthetic cinnabar which had to be retorted. Attempts at concentration by tables met with difficulty when it came to retorting the concentrates, because the elemental sulphur present formed an iron sulphide with the iron of the retorts.

PRESENT METHOD OF RECOVERY

The present method of recovery consists of the following operations:

1. Screening and sorting the low-grade ore to obtain a furnace feed containing 8 to 10 pounds of quicksilver per ton.
2. Treatment of the ore in a rotary kiln.
3. Precipitating most of the dust carried from the furnace in the gas stream.
4. Condensing mercury from the gas stream as (a) high-grade mud and (b) low-grade mud concentrates.
5. Flotation treatment of low-grade condenser mud to obtain high-grade concentrates.
6. Drying and retorting of high-grade condenser mud and high-grade flotation concentrates, making a finished product.

Screening and Sorting

Figure 1 gives the flow sheet of the screening and sorting plant and Table 1 gives details of equipment.

Of the 300 tons of run-of-mine ore fed per day, to the screening and sorting plant about 60 per cent is rejected as waste; of the remaining 40 per cent, or 120 tons, 43 tons per day is treated in the kiln and the balance of 77 tons per day stored in a stockpile. When the stockpile reserve has been built up to about 5,000 tons, the screening and sorting plant is closed down and the ore for the furnace is drawn from the stockpile until the accumulated supply is almost exhausted.

From the plant bin which has a capacity of 40 tons, the low-grade ore is fed to a trommel by an Allis-Chalmers apron feeder 3 feet wide on 7-foot centers, chain-driven by a 4-1/2-hp. motor, connected to a variable-speed, Link-Belt reduction unit. A swinging gate is installed above the discharge end of the feeder to prevent the large bowlders from hitting the trommel with too much of an impact. The trommel is 8 feet long and 42 inches in diameter and is made of 3/4-inch iron plate with 1-inch diameter punched holes. It is placed on a slope of 5° and is belt-driven by a 15-hp. motor.

The oversize product of the trommel passes onto an inclined 6-ply, rubber-surfaced picking belt, 36-inches wide, with 1/4-inch top cover and 1/32-inch bottom cover. The belt is 40 feet long, center to center, and is driven at a speed of 40 feet per minute by a 5-hp. motor. A series of water sprays wash the remaining fines from the coarse material and a sorter picks out the pieces of ore. About 20 tons of ore per month is sorted by hand from the picking belt. The reject from the picking belt goes to a stacking belt which carries

it to the waste dump.

The sorted ore from the picking belt is dropped into a 25-ton bin, and when this bin is full the ore is run through a 20 by 10 inch Blake-type crusher and crushed to minus 2 inches. The crushed product is conveyed by a travelling belt to the conveyer belt carrying the minus 1-inch undersize product of the trommel to the stockpile or furnace.

The fines washed from the picking belt are conveyed by launder to a Deister-Overstrom concentrating table. A shaking screen with 1/4-inch square holes removes wood from the fines before concentrating. The table produces concentrates containing about 1,000 pounds of quicksilver per ton. About 90 pounds of quicksilver is recovered by the table per month, the table operating daily, while the screening and sorting plant is in operation. The tailings from the table, which contain about 6 pounds of quicksilver per ton, are conveyed by launder to one of two storage ponds. When one storage pond is full, the table tailings are turned into another and the first pond is allowed to dry out. The tailings are eventually shoveled out of the pond into trucks and hauled to the stockpile.

The cinnabar concentrates, after drying, are retorted to recover the quicksilver. Retorting gives a higher recovery of quicksilver than furnace treatment, and it is for this reason that the fines washed from the picking belt are concentrated to recover part of the values for retort treatment before the material is sent to the storage ponds and thence to furnace treatment.

Furnace Treatment and Condensing Plant

Figure 2 gives a flow sheet of furnace treatment, dust precipitation, and condensing plant treatment. Table 1 gives the details of equipment used.

Furnace Treatment

When the screening and sorting plant is in operation the ore is conveyed by a travelling belt to the stockpile and thence by dragline, conveyor, and elevator to the furnace ore bin, which is of wood-stave construction and has a capacity of 50 tons.

From the furnace bin the ore is fed uniformly, at the rate of about 1.8 dry tons per hour, onto a traveling belt by a 30-inch Challenge feeder, and is discharged by the belt into the kiln. An electrical device, shown in Figure 3A, stops the belt and feeder if the ore is not being fed uniformly to the kiln. This device consists of two electrically connected contact straps which are suspended in the ore stream discharged by the feeder. The ore stream keeps the straps separated so that the electric circuit is open. If the ore stream is interrupted the straps come together, closing the electric circuit and starting a motor connected to a speedometer; at the same time an electric horn is sounded, which notifies the attendant, who may be elsewhere employed, that something is wrong with the ore feed. The speedometer, which is driven by a 1/32-hp. motor, registers the motor speed in miles per minute. The number of miles shown by the speedometer multiplied by the factor 0.54 gives the time in minutes that the belt was not running.

The speedometer and horn are also operated by closing the clutch switch or opening either the conveyor magnetic switch or contacts in the kiln feed hopper, as shown in Figure 3B. When the magnetic switch is opened, an extra arm clamped to its switch point axle makes contact with another point in the 110-volt speedometer and horn circuit.

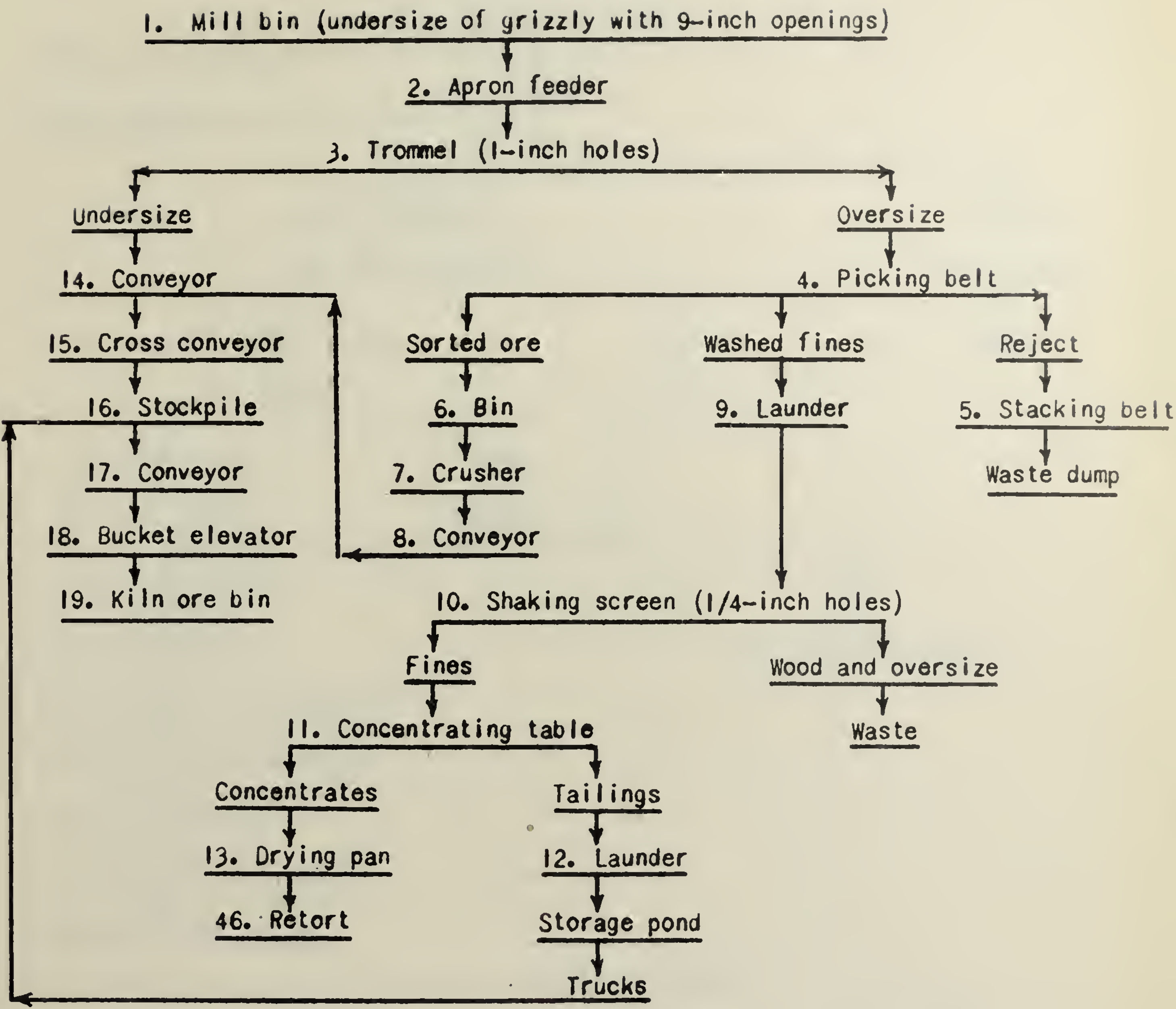
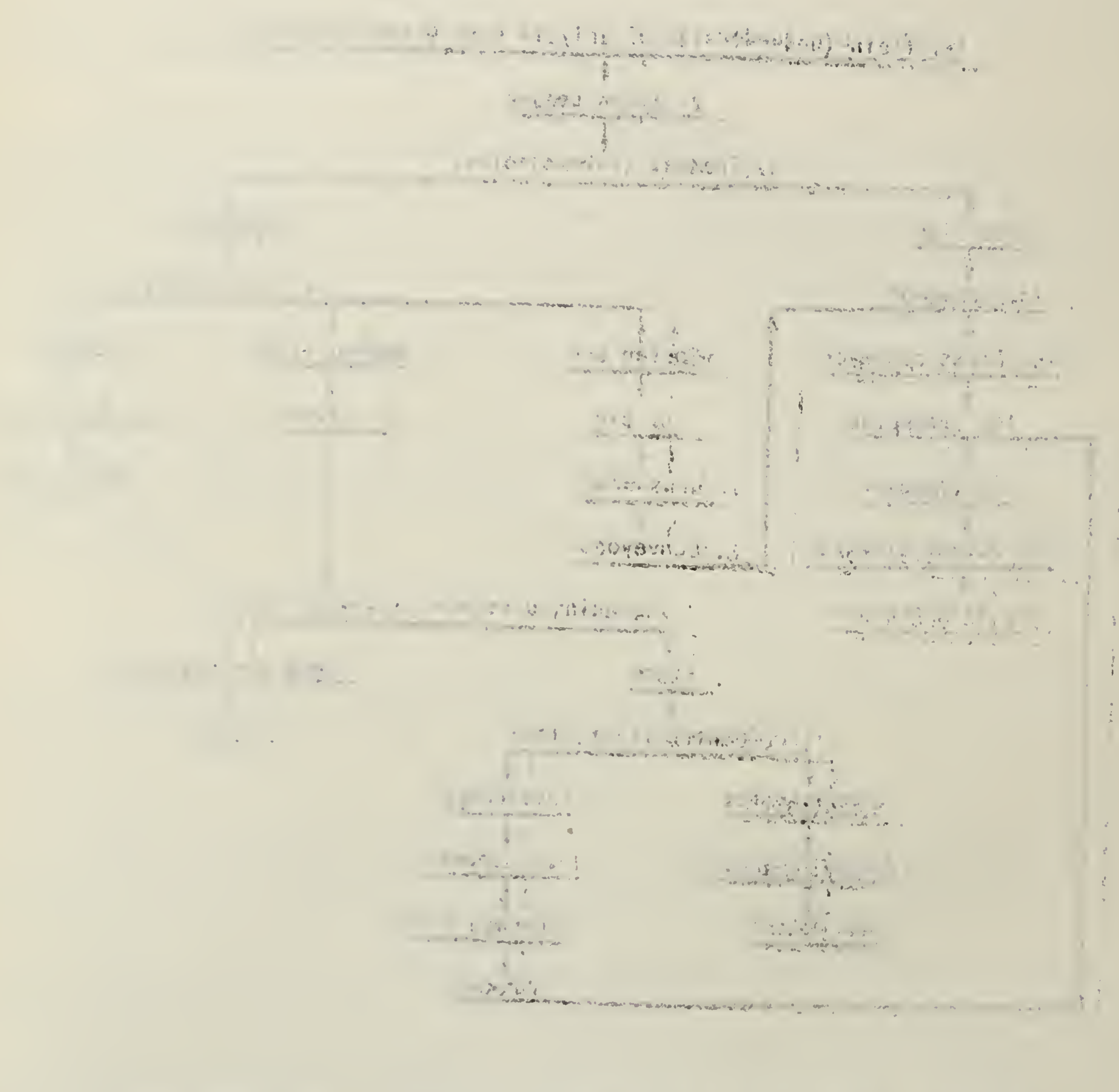


Figure 1.- Screening and sorting plant flow sheet. The numbers refer to details of equipment given in Table 1



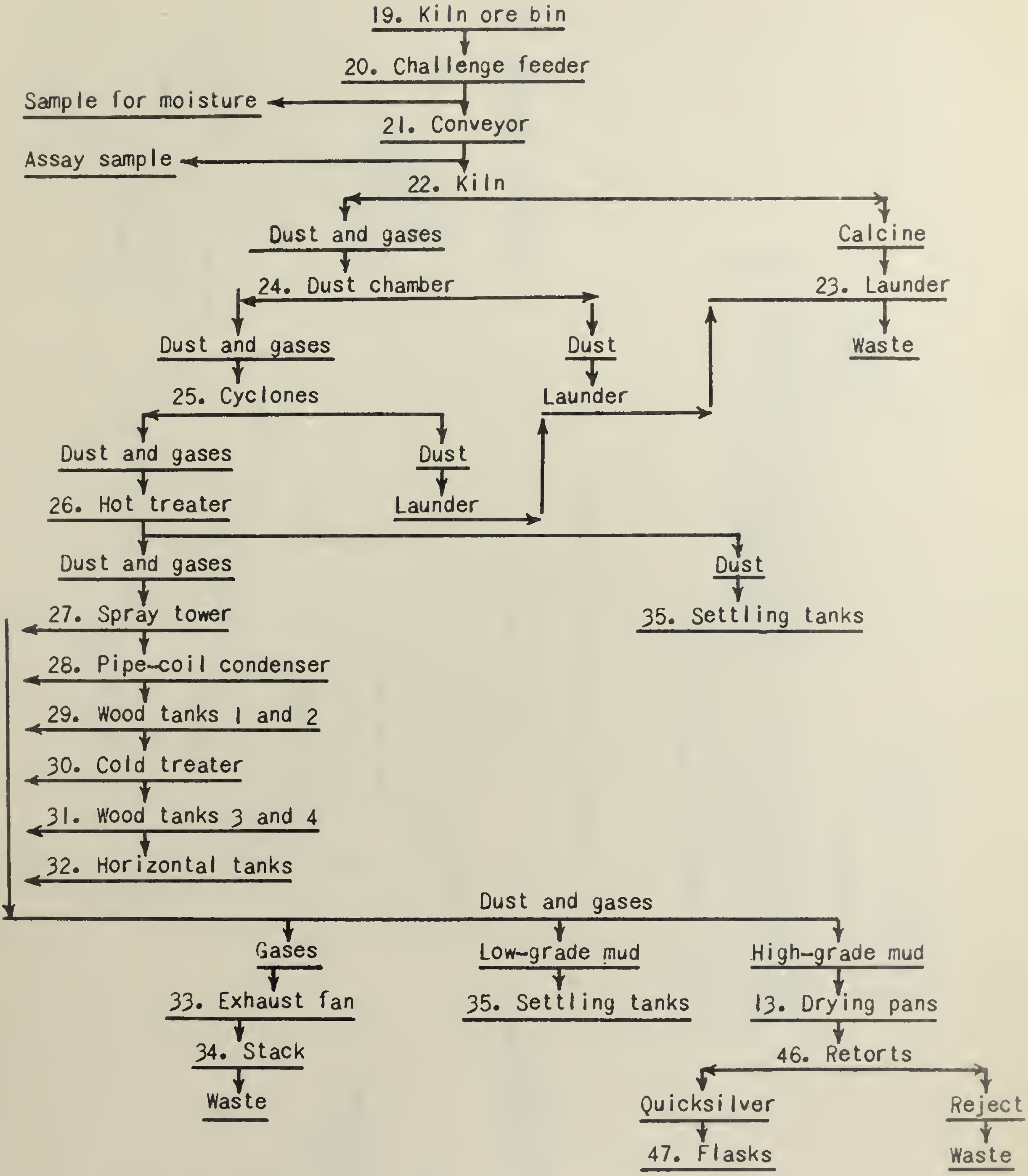
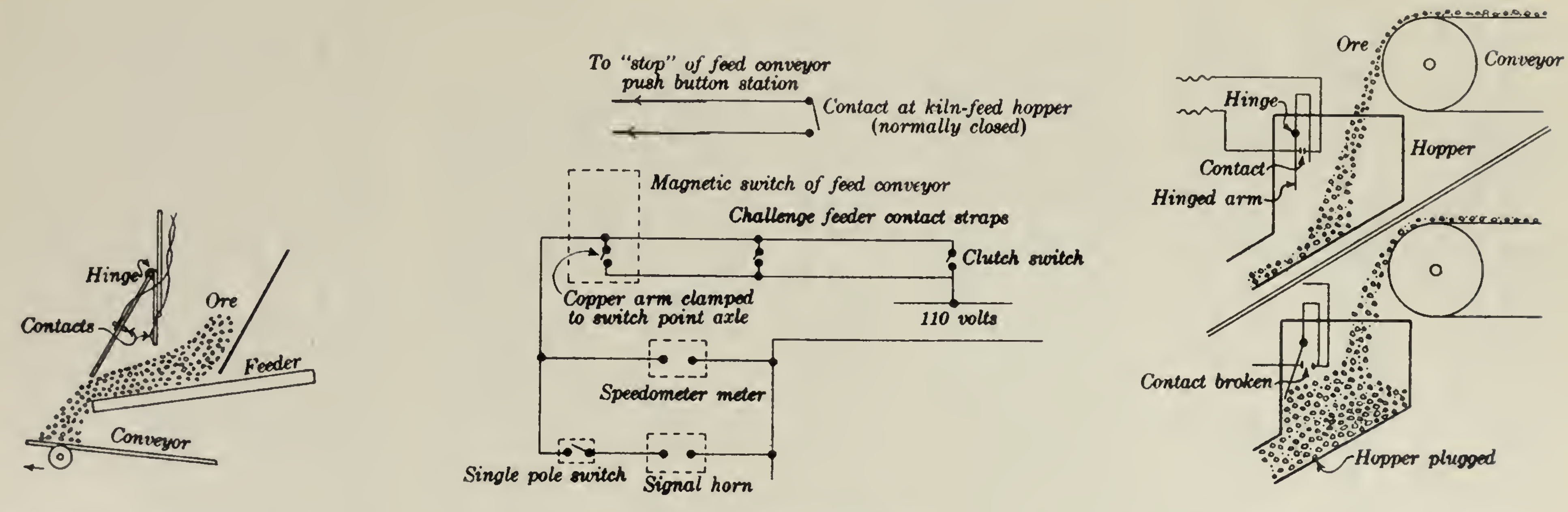


Figure 2.- Reduction and condensing plant flow sheet. Numbers refer to details of equipment given in Table 1

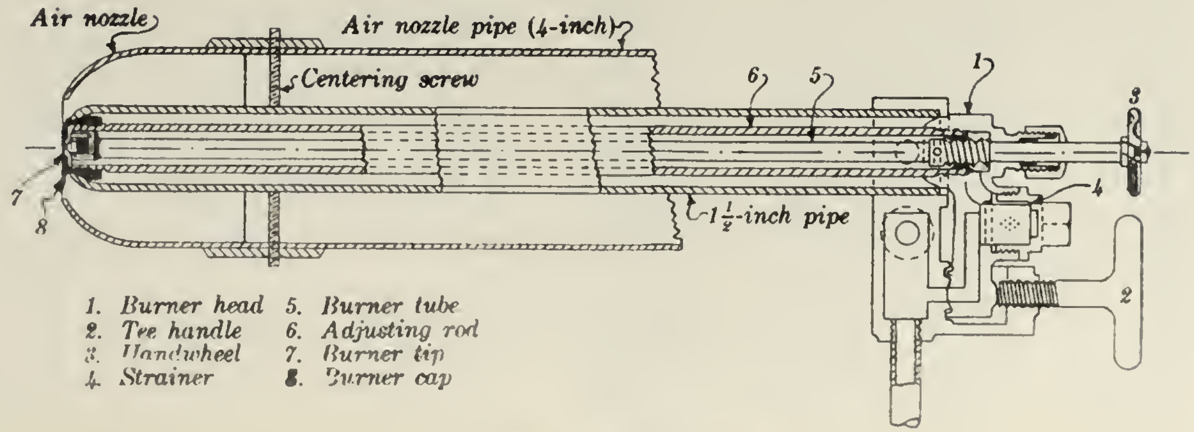


A.—SKETCH OF CONTACT STRAPS AT CHALLANGE FEEDER (SIDE ELEVATION)

B.—WIRING DIAGRAM

C.—SKETCH OF CONTACT AT KILN-FEED HOPPER

Figure 3.—Automatic kiln-feed shut-off system



1. Burner head
2. Tee handle
3. Handwheel
4. Strainer
5. Burner tube
6. Adjusting rod
7. Burner tip
8. Burner cap

Figure 4.—Ryder burner

The contacts in the kiln feed hopper are opened when the throat of the hopper is plugged, the ore building up against a movable hinged arm (see fig. 3C). Opening these contacts opens the magnetic switch, via the "stop" side of the push button station. A switch was cut into the horn circuit so that its operation could be avoided during shut-downs.

A sample of the ore is taken automatically at 5-minute intervals, as the ore drops from the conveyor belt into the furnace. The samples thus taken are combined into a composite sample for each shift.

In feeding the ore into the furnace, back-spilling is prevented by a set of six helical blades, 2 feet long, arranged around the inside of the kiln at the feed end. These blades help in carrying the ore away from the head of the kiln.

The rotary kiln, as first built and installed by the Pacific Car and Equipment Co., was 40 feet long and had an outside diameter of 5 feet and an inside diameter of 3 feet 10 inches. A 20-foot section was added later, making the kiln 60 feet in length. It was thought that by adding 20 feet to the length of the kiln the daily tonnage treated would be increased. Although the additional length failed to increase the capacity it proved more efficient in burning the sulphur out of the ore. The kiln has a slope of 1/2-inch per foot and is gear driven by a 7-1/2-hp., 900-r.p.m. motor.

The kiln lining consists of 2-1/2 inches of Diatex brick next to the shell and 4-1/2 inches of monolithic lining made of crushed fire brick and Lumnite cement in the proportion of 3 to 1, respectively. The materials required for lining the kiln are as follows:

Diatex bricks, size 2-1/2 by 4-1/2 by 9-inches, number of bricks	4,500
Black building paper, square feet	1,000
Lumber, size 1 by 6 inch by 10 foot, number of pieces	100
Tacks, pounds	8
Raw fire clay, cubic feet	20
Portland cement, sacks	25
Powdered infusorial earth, cubic feet	20
Lumnite cement, barrels	35
Crushed fire brick, cubic yards	15

The Diatex brick are chamferred on one side to fit the kiln shell. The mix of the mortar for the Diatex brick consists of 30 per cent by weight of raw fire clay, 40 per cent Portland cement, and 30 per cent of powdered infusorial earth. The mix for the fire brick concrete is 1 part by volume of Lumnite cement and 3 parts by volume of fire brick, crushed to minus 1-inch size with the coarse and fines taken together. The mix is used dry as possible and the concrete is well tamped behind movable wood forms. Building paper is used between the brick and the monolithic lining to prevent the brick from absorbing the moisture from the concrete. The forms are removed at the end of the first eight hours, and the concrete is sprinkled for four hours after that, so that it will not lose its moisture too rapidly.

The time required for lining the kiln is as follows:

	Hours
Allowing kiln to cool	24
Lining kiln	48
Setting	36
Burning wood in kiln to bring temperature up to 200°C.	8
Burning oil in kiln to bring temperature up to 625°C. (feeding small amount of ore)	8
Total time required to line kiln and bring into operation	124

The number of man-hours required for lining the kiln and bringing it into operation is 632.

The life of the lining is about two years.

As a means of securing complete oxidation of the free sulphur in the ore, the feeding of the ore and the firing of the kiln are done at the same end of the kiln as contrasted with the standard practice of countercurrent movement of ore and furnace gases, in which the kiln is fed and fired at opposite ends. By employing parallel firing and feeding at the proper temperature and with sufficient oxygen in the kiln, no unburned sulphur vapor is carried into the gas stream, consequently there is no tendency for the sulphur to recombine with the quicksilver in the condensing system. It was hoped that the sulphur content of the ore would average 10 per cent and thus render the fuel requirement low. In calorific value, 50 pounds of sulphur is equal to 1 gallon of oil. However, the average sulphur content of the ore has been about 2 per cent.

The fuel oil, which is filtered, is preheated to 95°C. by a U-shaped section of the pipe line extending about 8 feet into the furnace. The oil is atomized by a pressure of 75 pounds per square inch and is fed into the furnace through a Ryder burner, shown in Figure 4. The oil pressure pump and the fan for supplying the low-pressure air for shaping the flame are driven by a 3-hp. motor.

The temperature of the gases leaving the furnace averages about 625°C. The temperatures of the oil and of the furnace gases are recorded every hour by means of a recording pyrometer manufactured by the Wilson-Maeulen Co., Inc. A recording meter, made by the National Meter Co., registers the oil consumption. The furnace draft is recorded and averages about 0.18 inch of water.

Figure 5 shows a plan and Figure 6 a side elevation of the furnace plant, including the dust-collecting and flume-condensing equipment.

Dust Precipitation

The dust-precipitating equipment, as first installed, consisted of a large steel dust chamber containing checker brickwork lining at the end of the kiln, a set of 8-inch diameter cooling pipes placed horizontally in parallel, and a Cottrell precipitator or so-called "hot treater." This equipment was necessarily elaborate as it was known that the rotary kiln makes large quantities of dust and that the Sulphur Bank ore contained a large amount of fines. After the plant was put into operation the amount of dust exceeded the estimated amount, and for that reason this section of the plant has undergone a number of changes. The amount of dust caught by the Cottrell precipitator was less than 50 per cent

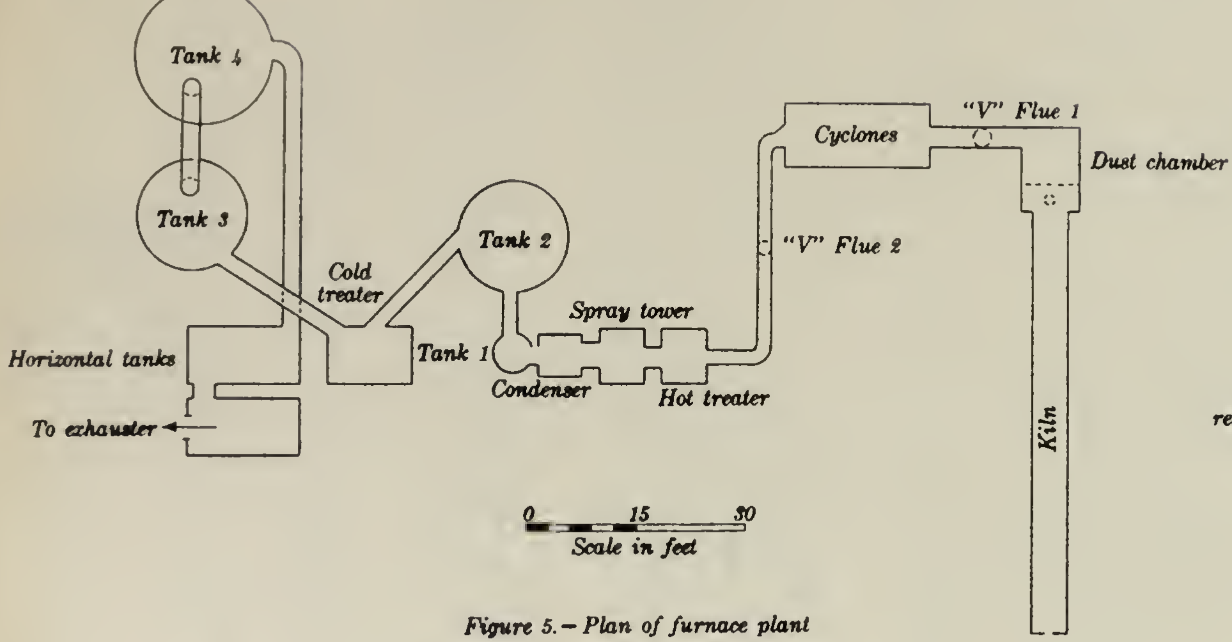


Figure 5.– Plan of furnace plant

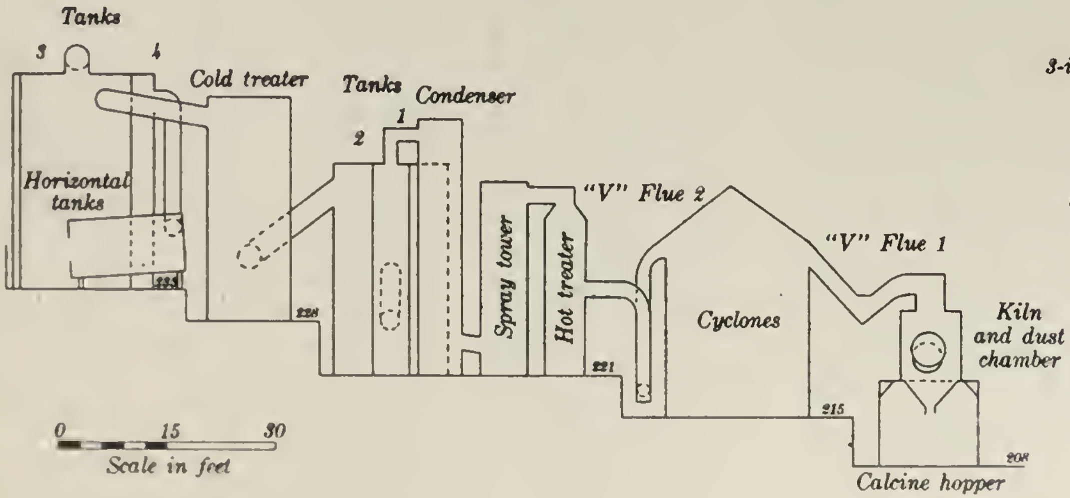


Figure 6.– East side elevation of furnace plant

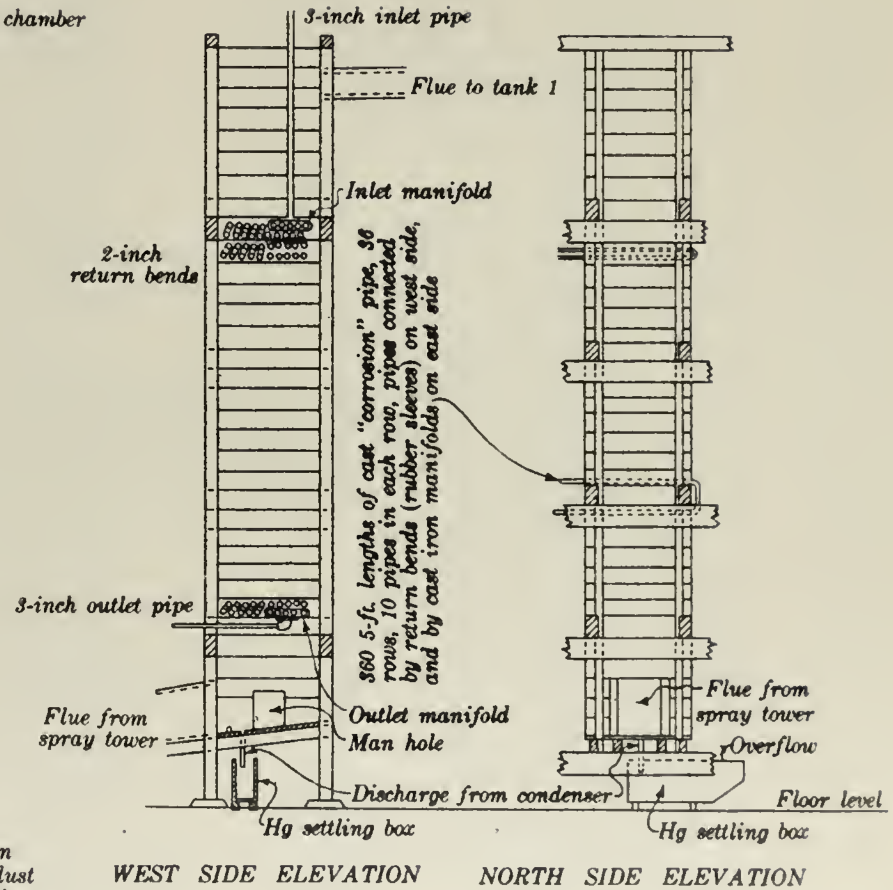


Figure 7.– Pipe coil condenser

of the total dust reaching it and the remainder caused a dust problem in the condensing system. The checker brickwork in the steel dust chamber and the horizontal cooling pipes choked up with dust so as to cause a loss of draft.

The dust-precipitating equipment, as it is now operating, consists of the steel dust chamber operating with a chain curtain instead of the former checker brickwork; 12 cyclone dust precipitators, in two sets of 6 each, the sets operating in parallel and replacing the cooling pipes formerly used; and the Cottrell precipitator. The 12 cyclones, each 24 inches in diameter, are so arranged that they can be placed partly in series. As the dust precipitating system is operated at present, a cooling of the gas stream takes place in the cyclones equal to that formerly afforded by the cooling pipes, the draft loss is reduced to a minimum, and most of the dust is precipitated. The amount of dust carried into the condensing system amounts to about 1 ton per day.

The temperature of the gases as they leave the cyclones averages about 235°C. The temperature is taken at the V-shaped flue connecting the cyclones with the Cottrell precipitator and is recorded every hour by an attendant.

Condensing

The condensing system as first installed in 1927, consisted of a brick spray tower, a wood condensing chamber with internal water coils, a Cottrell precipitator or so-called "cold treater," two circular wood-stave tanks placed horizontally, a steel exhaust fan enclosed in a wooden chamber, and a wood-stave stack 48 inches in diameter and 40 feet high. After this plant was in operation a short time, stack tests were made to determine the amount of quicksilver being lost in the gases expelled from the stack, and it was found that this loss amounted to 100 pounds of quicksilver per day. Additional units, consisting of four large circular tanks, were added to the condensing system as soon as possible. Subsequent stack tests made after the additional units were installed have shown the stack loss to be less than 10 pounds of quicksilver per day.

Each of the six wooden condensing tanks has a wooden partition which allows the gases to circulate through both halves of the tank. The bottom of each tank is inclined three-fourths inch per foot. By-pass arrangements are provided for each pair of tanks so that the monthly clean-up can be made without closing down the entire plant. During the warm weather the vertical wood tanks are kept wet on the outside. The water is piped to the drum head at the top of each tank, from where it overflows and runs down the sides.

The spray tower is equipped with a water-sealed outlet at the bottom through which the low-grade mud is drawn off continuously and carried by a launder to settling tanks for flotation treatment. The average flow of spray tower and condenser water is 20,000 gallons per day.

The water-coil condenser unit, shown in Figure 7, contains 360 Corrosiron pipes, each 5 feet in length and having an outside diameter of 2-1/2 inches.

A partial analysis of the Corrosiron pipe after one year's use was as follows:

Metallic mercury	Trace
Combined mercury	1.80 per cent
Sulphur	17.46 do.
Iron	23.60 do.
Insoluble	41.20 do.

The average life of the Corrosiron pipe is about two years. An idea of the acidity of the gas stream can be gathered from the following test, recently made. A piece of common steel, suspended in one of the flues for five days, lost 0.116 per cent of its weight per day due to corrosion.

A steel fan was originally used for exhausting the gases, but its blades gave way and had to be replaced in nine months because of corrosion. The fan in present use is 100 inches in outside diameter and is made of Monel metal. It is direct-connected to a 25-hp. motor and operates at 500 r.p.m., exhausting the gases from the system at a water-gage pressure of 4-1/2 inches.

The volume of gases passing through the stack is about 6,500 cubic feet per minute. The mercury content in the gases is approximately 0.445 milligrams per cubic foot.

The draft at the inlet of the water coil condenser is 1.5 inches of water.

Flotation

The treatment of the low-grade mud from the condensing system, as first designed in 1927, was simple but proved to be inefficient. A series of launders, boxes, and riffles, together with a settling tank were used to catch the low-grade mud. The overflow from the settling tank was collected in a pond, and after settling and drying the mud was returned to the kiln with most of the mud caught in the launders, boxes, riffles, and settling tank. The mud collected was much greater in amount than anticipated and was of low mercury content. Treatment of the mud by drying and retorting was tried, but the amount greatly exceeded the capacity of the three D-retorts.

The problem of concentrating the mud to retort grade was satisfactorily solved by the installation of two additional settling tanks, a tank with agitating equipment and a two-cell Kraut flotation machine with auxiliary filtering apparatus.

In the present flotation flow sheet, shown in Figure 8, the low-grade mud from the condensing system flows by launder to either of two wooden settling tanks, each 16 feet in diameter and 5 feet high. The overflow from the tanks is conveyed by a launder to a third settling tank. The overflow from this third settling tank is sent to waste. A sample of the waste overflow is taken automatically every two minutes. Daily composite weights and assays of the solids in the overflow product show a quicksilver loss of less than 5 pounds per day.

The mud from the three settling tanks is collected into the sumps from where it is pumped by 2-inch Wilfley pumps to a tank equipped with a Devereux agitating machine, rotating at a speed of 25 r.p.m. From the agitator the pulp goes to the two-cell Kraut flotation machine in which the cells arranged in series. The pulp is circulated through the flotation cells about five times. When panning of the froth shows little mineral of value, the tailings are sent to waste.

The flotation heads average about 74 pounds of quicksilver per ton; the concentrates about 600 pounds per ton; and the final tailings about 2 pounds per ton. Pulp density is approximately 20 per cent of solids by weight.

The flotation reagents and amounts used per ton of original ore treated are as follows:

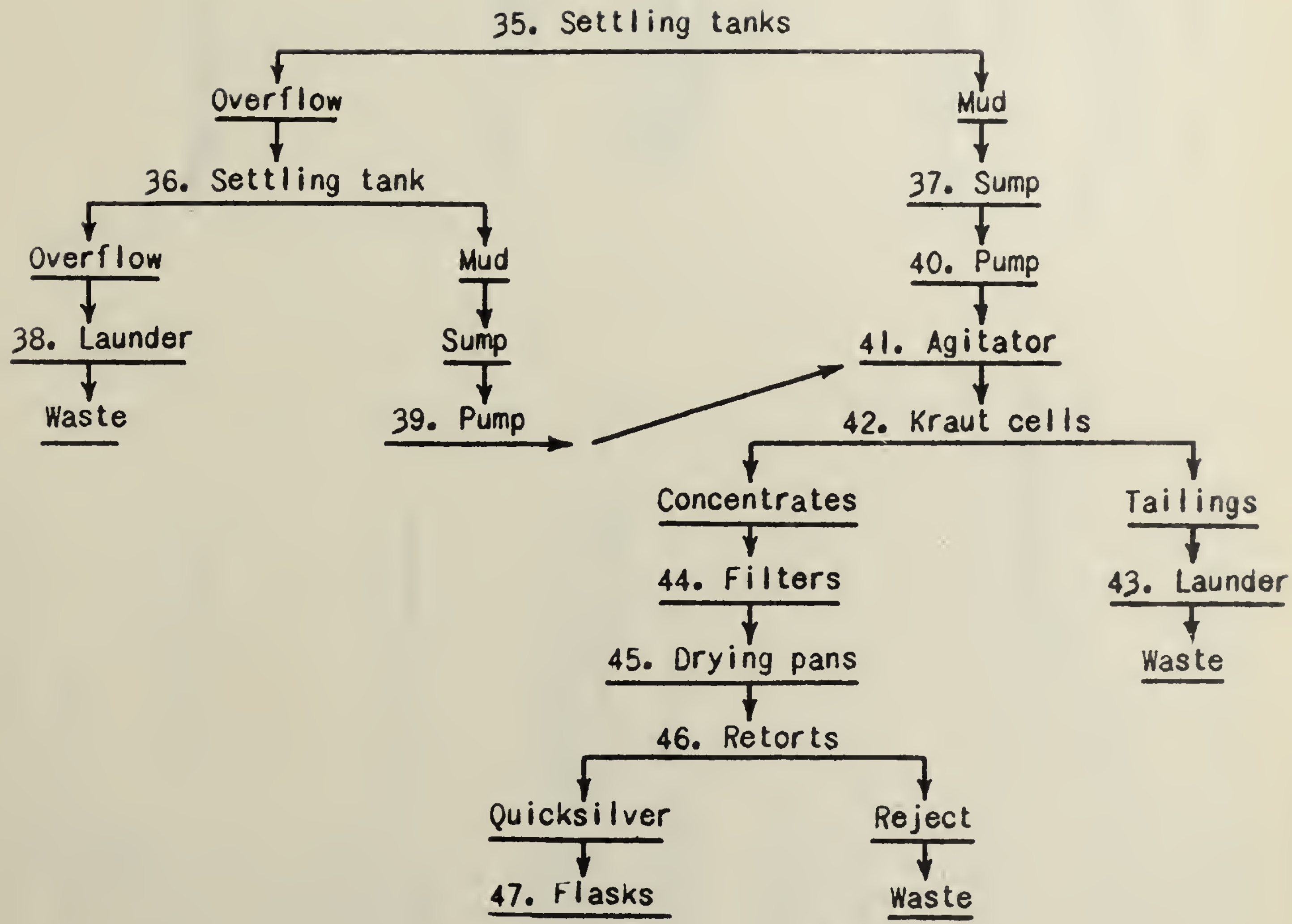
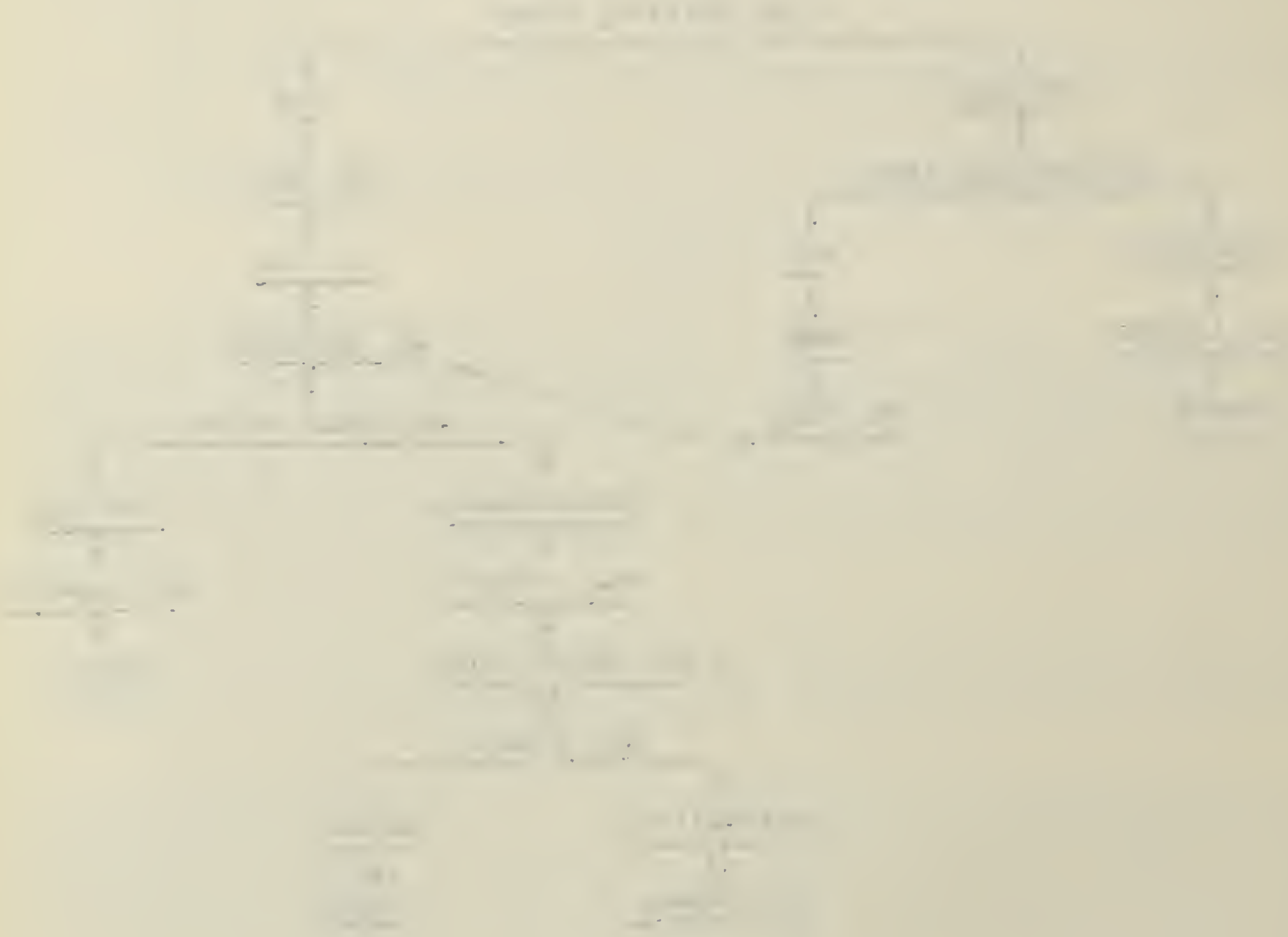


Figure 8.- Flotation flow sheet. Numbers refer to details of equipment given in Table I



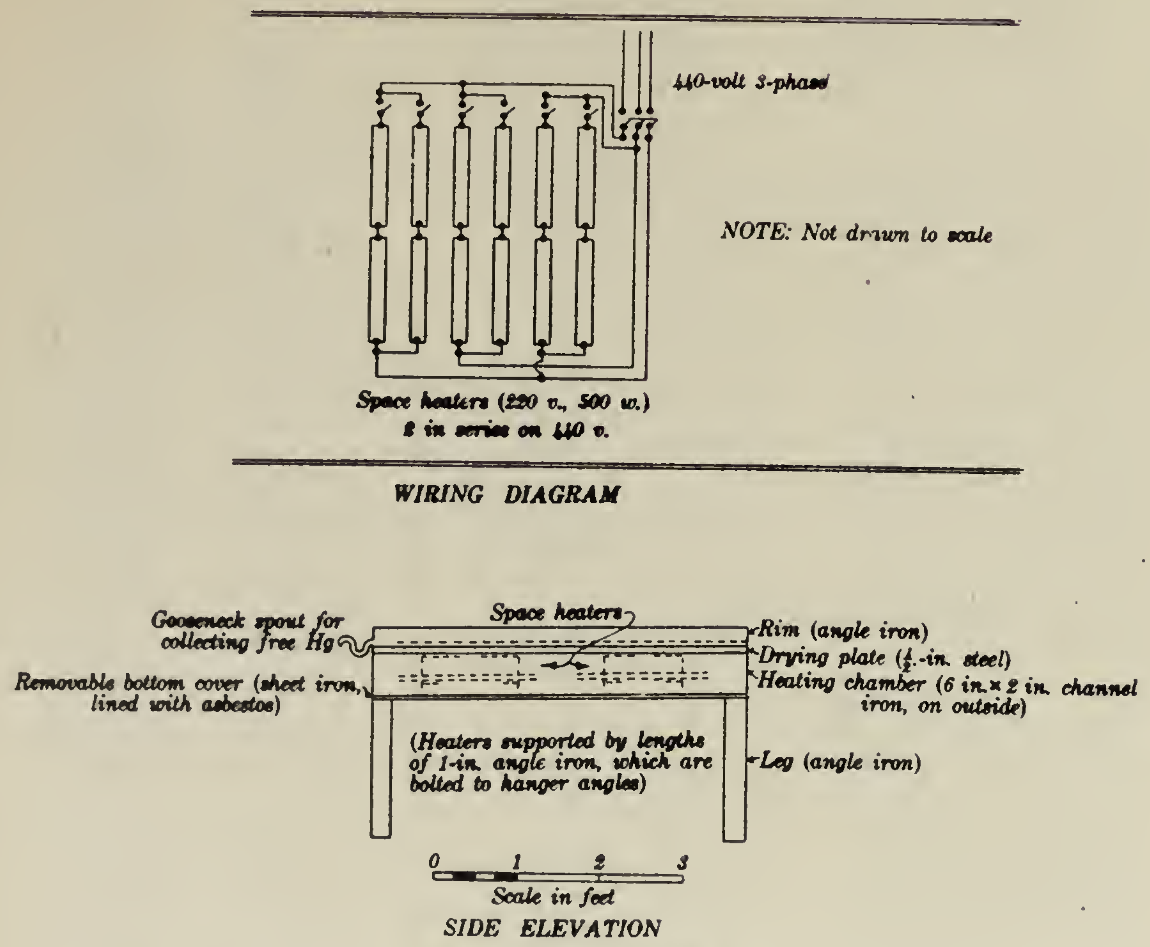


Figure 9.—Electric drying pan

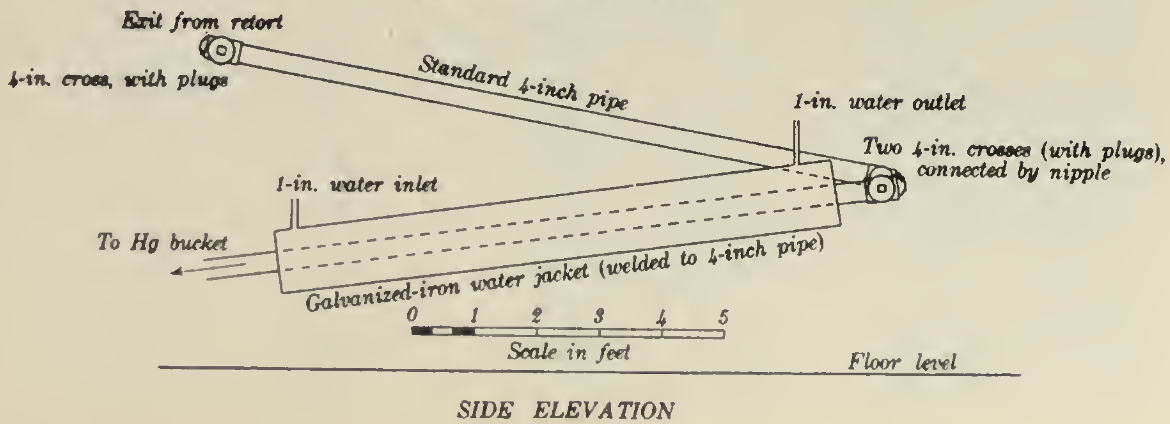


Figure 10.—Retort condenser

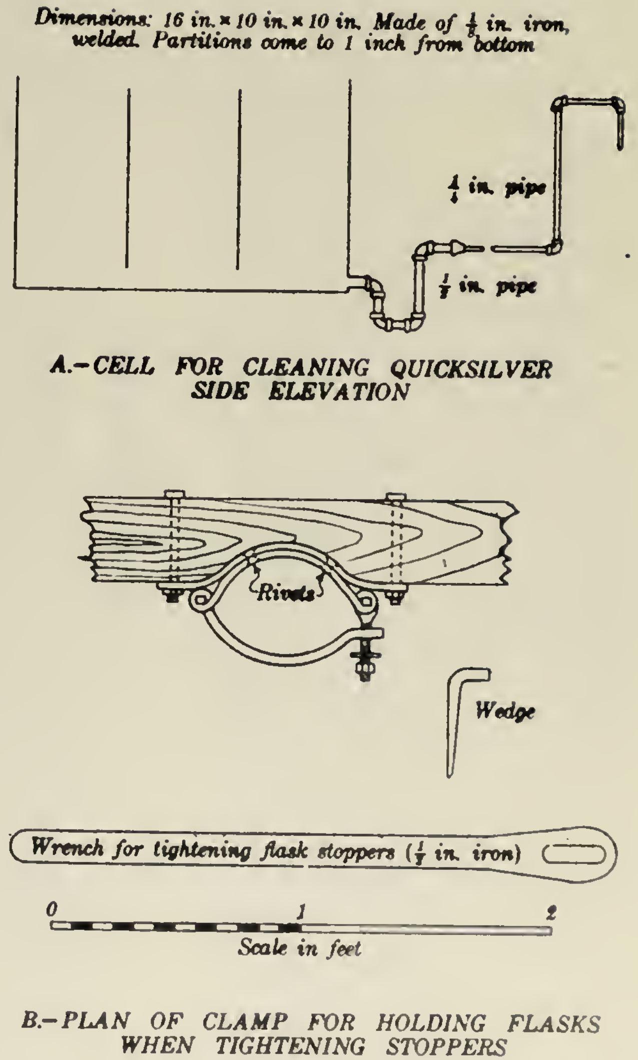


Figure 11.—Cleaning cell and flask clamp

	Pounds per ton
Sodium aerofloat	0.0543
Pine oil	0.3093
Barrett No. 4	0.0186

The flotation concentrates go to two drum filters made from 50-gallon oil drums and arranged in series. Filtering medium consists of 12-ounce duck placed over a screen with 1/4-inch holes. The screen in turn is placed over iron rods. The canvas filter cloths are changed every three days. The vacuum for dewatering the concentrates is supplied by an Oliver vacuum pump, belt driven by a 5-hp. motor. The vacuum is maintained at 25 inches of mercury.

Drying and Retorting

The dewatered flotation concentrates are dried in electrically heated drying pans (fig. 9). The concentrates are mixed with quicklime and sprinkled with water; the heat generated by the slacking of the lime aids the drying operation, and the lime added is also needed as a flux in the retorting of the dried concentrates product. The time required to dry a charge of about 325 pounds is eight hours.

After drying, the concentrates are retorted in three standard D-retorts. A working drawing of such a retort can be found on page 144 of Duschak and Schuette's "Metallurgy of Quicksilver," Bureau of Mines Bulletin 222. A retort charge weighs about 300 pounds and is left in the retort for a period of 24 hours. Retort residues mixed with water are used to seal the retort doors. Each retort is equipped with a Ray oil burner which has independent air and oil control. The condenser pipe leading from each retort is water-jacketed as shown in Figure 10. At the front of each retort, over the door, is a hood through which the retort gases can be drawn by means of a suction fan when the door is opened.

A cell for cleaning the quicksilver is shown in Figure 11A. The cell contains water, into which the quicksilver is poured. The quicksilver is cleaned by passing through the water, and is drawn from the cell through a 1/2-inch pipe nipple, welded close to the bottom of the cell. This nipple leads into a jointed pipe line with a series of bends; the pipe can be turned up or down to various levels, thus controlling the flow of the quicksilver. The pipe discharges into the weighing bucket which, when containing the correct amount of quicksilver is emptied into a flask. A device for holding the flasks while tightening the stoppers is shown in Figure 11B.

SAMPLING, ASSAYING, AND CONTROL OF OPERATIONS

In furnace treatment and subsequent operations the following products are sampled and assayed for mercury daily: Kiln feed, kiln calcine or residue, cyclone dust, low-grade mud, retort residue, flotation heads, concentrates, and tailings.

Mechanical samplers are used for the kiln feed, low-grade mud and flotation tailings. The stream of the first-named product is cut every five minutes by the sampler and the stream of the other two products every three minutes. The retort residue product is sampled by hand once each day and the remaining samples are taken by hand every hour.

In addition to the sample of kiln feed taken for assay purposes, two other kiln feed samples are taken, one of which is used for the computation of kiln feed tonnage and

the other for the determination of moisture content. The sample for tonnage computation is taken at the Challenge feeder which discharges onto the kiln feed conveyor. The kiln-feed ore stream is cut and collected for 10 seconds once every hour. Each sample is weighed, and the weights are averaged for the day. Kiln-feed tonnage is computed from the average sample weight. At the time of taking the sample for tonnage computation, another grab sample is taken and put into a jar for determination of moisture content.

The amount of low-grade mud which is lost each day in the waste water is determined by timing the filling of a 5-gallon container once per hour and determining the solids in the water. The determination of solids is made by filtration of 3,065 cubic centimeters of the waste water taken from the sample obtained from the automatic sampler. The solids are weighed and the loss of low-grade mud is computed for the day.

The method of assaying in use at the Sulphur Bank plant was developed by the Bureau of Mines and is described in their Technical Paper 227.[3] Briefly, it is known as the "glass-tube method," and consists in separating the quicksilver by distillation in a closed tube, dissolving it in nitric acid, and titrating with potassium thiocyanate.

The sulphur is determined by the standard barium chloride method.

Once every four months a "stack test" is made to determine the daily amount of quicksilver expelled into the atmosphere. The apparatus used for this test can be set up at the stack proper, but for convenience is usually installed at the flue between condensing tank No. 4 and the horizontal tanks. This apparatus described in the order of flow consists of the following parts: A sampling tube inserted into the flue, a water-jacket condenser, a filter tube containing glass wool, a filter flask, a gas meter with draft gage and thermometer connected, and a jet. The mercury and water in the gas condense and drop into the condenser. The quicksilver is caught by the glass wool and the water goes into the flask. Calculations according to the following formula are made from the data secured by the above apparatus and readings from a thermometer and draft gage connected to the flue.

$$\text{Quicksilver lost per 24 hours, pounds} = \frac{\text{Milligrams of mercury in sample} \times \text{Total volume of gas leaving flue, cubic feet per minute} \times 3.17}{\text{Volume of gas metered for sample, cubic feet} \times 1{,}000}$$

The factor 3.17 is the product of the factors 0.0022, which changes grams to pounds, and 1,440, the number of minutes in 24 hours.

The assayer fills out a "daily sheet" which gives the wet weight of ore in tons, the per cent of moisture, the dry weight of ore in tons, fuel oil consumed in gallons per wet ton of ore and time lost in minutes. Figures for these items are given for each shift and are also totaled and averaged for the day. This sheet also gives weights and assays of kiln feed, low-grade mud, and flotation tailings; assays only of kiln residue, cyclone dust, flotation heads, and concentrates products; and the amount of quicksilver produced by pipe-coil condenser, retorts, and clean-ups. Retort production of mercury is itemized as follows: Condenser mud, flotation concentrates, and table concentrates.

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3 - Bouton, C.M., and Duschak, L.H., The Determination of Mercury: Tech. Paper 227, Bureau of Mines, 1920, 44 pp.

For this daily sheet the assayer secures part of the information from the "daily kiln report," which is made out by the furnace-plant operators. A section of a typical daily kiln report would appear as follows:

Section of daily kiln report

Hour	Temperature, °C.						Drafts, inches of water			Color	
	Kiln	V flue No. 2	Condenser In	Condenser Out	Stack	Oil	Kiln	Condenser	Total	Mud	Kiln residue
7 a.m.	625	235	70	41	17	95	0.18	1.50	2.40	Gray	Gray

The operators fill in the above items hourly, and in addition give reasons for any shut-downs which might occur. The fuel-oil meter reading is put on this sheet once each shift.

It can be seen that the filling in of the daily kiln report requires the reading of a number of instruments, and the plant is well equipped in this respect. The kiln and oil temperatures are obtained from an indicating pyrometer, as previously described, with the Pyod thermocouples inserted in the dust chamber and in the intake of the oil preheating coil. The other temperatures are read from full immersion chemical thermometers. The kiln draft is indicated by a differential draft gage, with the tube inserted in V flue No. 1. The other drafts are read from standard U gages. The oil consumption is read from the previously described oil meter calibrated for 27° fuel oil.

The items "mud color" and "residue color" are filled in from visual observation, and are simply put down as a check on the instrument readings. When the color is gray, or any light color, conditions are satisfactory and little quicksilver is escaping.

Table 1.—Details of equipment

Note: The following numbers correspond to those given in the flow sheet diagrams of Figures 1, 2, and 8.

1. Inclined-bottom wooden bin, capacity 40 tons of wet ore, slope 45°.
2. Allis Chalmers apron feeder, 3 feet wide on 7-foot centers, chain-driven by 4-1/2-hp., variable-speed motor connected to link belt reduction unit.

3. Trommel, 8 feet long and 42 inches in diameter, slope 5°, made of 3/4-inch iron plate with 1-inch punched holes, belt-driven at speed of 18 r.p.m. by 15-hp. motor.

4. Picking belt inclined 15°, 36 inches wide, 6-ply, 32-ounce duck, rubber surfaced, 1/4-inch top cover, 1/32-inch bottom cover, on 40-foot centers, speed 40 feet per minute, belt-driven by 5-hp. motor.

5. Stacking belt inclined 20°, 30 inches wide, 6-ply, 32-ounce duck, rubber surfaced, 1/4-inch top cover, 1/32-inch bottom cover, on 150-foot centers, speed 150 feet per minute. Water spray on under side of belt at discharge end to wash off adhering fines. Belt-driven by 5-hp. motor.

6. Inclined-bottom wood bin with rack and pinion gate, 25 wet tons capacity; slope, 40°.

Table 1.--Details of equipment (Continued)

7. Allis Chalmers 20 by 10 inch Blake-type crusher with manganese steel wearing plates. Belt-driven by 20-hp. motor.

8. Belt conveyor, 18 inches wide, 4-ply, 28-ounce duck, rubber surfaced, 1/8-inch top cover, 1/32-inch bottom cover, on 37-foot centers, speed 180 feet per minute. Belt-driven by a 3-hp. motor.

9. Box launder made of 1-inch boards, 4 by 6 inches in cross section, slope 7/8-inch per foot.

10. Shaker screen, 1/4-inch holes, belt-driven by 3-hp. motor.

11. Deister-Overstrom concentrating table, 6-1/2 by 14-1/2 feet, belt-driven by 5-hp. motor.

12. Wood launder made of 1-inch boards, V shaped, 12 by 12 inches in cross section

13. Electric drying pans, 4.5 by 4.5 feet, sloping in two directions, nichrome heating elements.

14. Conveyor belt, 18 inches wide, 4-ply, 28-ounce duck, rubber surfaced, 1/8-inch top cover, 1/32-inch bottom cover, on 340-foot centers, speed 180 feet per minute, belt-driven by 3-hp. motor.

15. Cross conveyor belt, 18 inches wide, 4-ply, 28-ounce duck, rubber surfaced, 1/8-inch top cover, 1/32-inch bottom cover, on 38-foot centers, speed 180 feet per minute, belt-driven by 3-hp. motor.

16. Stockpile capacity about 5,000 dry tons.

17. Conveyor belt, 18 inches wide, 4-ply, 28-ounce duck, rubber surfaced, 1/8-inch top cover, 1/32-inch bottom cover, on 48-foot centers, speed 180 feet per minute, belt-driven by 3-hp. motor.

18. Bucket elevator, 47 feet long between centers, speed 190 feet per minute, chain-driven by 7-1/2 hp. motor. Buckets, 15 inches long and 7 inches wide, spaced on 18-inch centers.

19. Kiln ore bin, flat bottom, wood-stave construction, capacity 50 tons of wet ore.

20. Challenge feeder, 30 inches in diameter, chain-driven by 3-hp. motor.

21. Conveyor belt, 18 inches wide, 4-ply, 28-ounce duck, rubber surfaced, 1/8-inch top cover, 1/32-inch bottom cover, on 65-foot centers, speed 108 feet per minute, belt-driven by 3-hp. motor.

22. Kiln 60 feet long, 5 feet outside diameter, 3 feet 10 inches inside diameter, slope 1/2-inch per foot, gear-driven by 7-1/2 hp. motor. Lined with 2-1/2 inches of Diatex brick and 4-1/2 inches monolithic fire brick concrete. Equipped with Ryder burner.

Table 1.—Details of equipment (Continued)

23. Launder, V shape, made of 1-inch boards 12 inches wide. Slope of launder, 6 inches per each 16 feet. Calcine discharges into water flowing in the launder.

24. Steel dust chamber, 12 feet long, 10 feet high, 8 feet wide, equipped with chain curtain. Discharge from dust chamber through V-shaped flue made of steel pipe 30 inches in diameter.

25. Cyclone dust precipitators, 12 in number, grouped in two sets of six each, the groups arranged in parallel.

26. Hot treater, cver all dimensions, 25 feet high and 14 by 5 feet in cross section.

27. Spray tower, height 26 feet, 7 by 6 feet in cross section. Water for sprays pumped by 2-3/4 by 4-inch Fairbanks Morse duplex plunger pump, belt-driven by 10-hp. motor.

28. Pipe coil condenser, height 33 feet, 4 by 4-1/2 feet in section, contains 360 Corrcsiron pipes, each 5 feet in length and 2-1/2 inches outside diameter.

29. Circular wood-stave tanks, 2 in number, one 30 feet high, 5 feet in diameter, and the other 30 feet high and 16 feet in diameter.

30. Cold treater, 16 feet high, 6 by 10 feet in section.

31. Circular wood-stave tanks, 2 in number, one 30 feet high and 20 feet in diameter, and the other 30 feet high and 16 feet in diameter.

32. Circular wood-stave tanks placed horizontally, each 16 feet long and 8 feet in diameter.

33. Exhaust fan, enclosed in wooden chamber 100 inches in outside diameter. Fan is of Monel metal.

34. Stack, 4 feet in diameter, and 40 feet high, made of wood staves.
35. Wood-stave tanks, 2 in number, each 16 feet in diameter and 5 feet high.
36. Settling tank, 20 feet in diameter and 5 feet high.
37. Concrete sump.
38. Wood launder made of 1-inch boards, 12 by 12 inches in cross section.
39. Wilfley pump, 2-inch centrifugal, direct-connected to 5-hp. motor.
40. Same as item 39.

41. Agitator tank, 10 feet in diameter and 10 feet high. Devereux agitating mechanism driven at 25 r.p.m. by 5-hp. motor. Belt and pinion drive.

42. Two Kraut flotation cells driven by 5-hp. motor with texrope drive.

Table 1.—Details of equipment (Continued)

43. Launder made of 1-inch boards, cross section of launder 6 by 8 inches.

44. Drum filters made out of 50-gallon oil barrels, canvas filter cloth laid on screen with 1/4-inch holes, which in turn is placed on iron rods. Vacuum 25 inches of mercury maintained by Oliver vacuum pump, belt-driven by 5-hp. motor.

45. Same as 13.

46. Standard D-retorts, 3 in number, with water-jacketed condensing pipe; size of retort fire box 2 by 2 feet in cross section and 9 feet long; retort size 1 by 1-1/2 by 8 feet long. Each retort equipped with Ray oil burner.

47. Iron flasks, each holding 76 pounds 1 ounce of quicksilver.

Table 2 gives metallurgical data sheet and Table 3 the metallurgical balance sheet for the month of April, 1930.

Table 4 gives the wage scale at the plant; table 5 gives a summary of plant costs, and Table 6 presents plant costs in units of labor, power, and supplies.

Table 2.—Metallurgical data for month of April, 1930

Dry ore treated, tons	1,290
Quicksilver produced, pounds	9,118.2
Average quicksilver content per ton of ore, pounds	8.32
Recovery of quicksilver, per cent	84.95
Oil used per dry ton of ore, gallons	22.8
Moisture in ore sent to mill, per cent	30
Temperature of kiln, °C.	625
Temperature in flue discharging from cyclones, °C.	235
Temperature at pipe coil condenser inlet, °C.	70
Temperature at pipe coil condenser outlet, °C.	41
Temperature at stack, °C.	17
Draft at kiln, inches of water	0.18
Draft at pipe coil condenser inlet, inches of water	1.50
Draft at end of condensing systems, inches of water	2.40

Table 3.—Metallurgical balance sheet for the month of April, 1930

	Weight, dry tons	Assay, pounds quicksilver per ton	Weight of quicksilver, pounds
Screening and sorting plant:			
Low-grade ore mined[1]	9,000	4.50	40,500
Waste rejected	5,400	1.50	8,100
Ore sorted and screened	3,600	9.00	32,400
Ore stored in stock pile	2,310	9.38	21,667
Ore sent to furnace	1,290	8.32	10,733
Furnace plant losses of quicksilver:			
Kiln residues, tonnage estimated to be 95 per cent of feed	1,225.0	0.40	490.0
Cyclone dust, tonnage estimated	45.0	2.50	113.0
Flotation tailings	40.0	2.00	80.0
Low-grade mud	2.1	21.43	45.0
Retort residues, tonnage estimated	18.0	0.40	7.0
Stack losses, estimated	-	-	300.0
Unaccounted for	-	-	579.8
Total loss of mercury	-	-	1,614.8
Furnace plant production of mercury:			
From condenser	-	-	5,107.2
From retorting of mud	-	-	1,731.0
From retorting of table concentrates	-	-	91.2
From retorting flotation concentrates	-	-	1,094.4
From monthly clean-up	-	-	1,094.4
Total production for month	-	-	9,118.2
Distribution of monthly clean-up mercury:			
Horizontal tanks	-	-	87.5
Tanks 3 and 4	-	-	54.7
Cold treater	-	-	207.9
Tanks 1 and 2	-	-	404.9
Spray tower and condenser	-	-	186.1
Agitator	-	-	153.3
Total	-	-	1,094.4

1 - Wet weight of ore, 12,857 tons. Assay content of wet ore 3.15 pounds quicksilver per ton.

Table 4.--Wage scale

	Per 8-hour shift
Screening and sorting operations:	
Grizzly man, skip loader, and plant helpers	$4.50
Hoistman and plant operator	5.50
Furnace, flotation, retort, condenser, and miscellaneous operations:	
Plant operators	5.50
Plant helpers and general labor	4.50
Retort and drag-line operator	5.00
Carpenter	6.00
Blacksmith and electrician	6.50
Assaying:	
Assayer	5.00

Table 5.--Summary of costs per ton of ore treated for April, 1930

Tons of dry ore treated: 1,290. Pounds of quicksilver produced: 9,118.2.

	Labor	Power	Oil	Reagents	Other supplies	Total
Screening and sorting[1]	$0.594	$0.135	–	–	$0.106	$0.835
Furnace treatment	.138	.018	$1.368	–	.185	1.709
Flotation	.246	.029	–	$0.044	.068	.387
Retorting	.031	.016	.073	.029	.028	.177
Assaying	.078	.008	–	–	.021	.107
Condensing	.315	.034	–	–	.185	.534
Miscellaneous	.116	[2].174	–	–	.069	.359
Total	$1.518	$0.414	$1.441	$0.073	$0.662	$4.108

1 - Includes crushing of the ore sorted on picking belt.

2 - Includes water pumping, exhausting furnace gases, and conveying.

Table 6.—Summary of costs in units of labor, power, and supplies for April, 1930

Tons of dry ore treated: 1,290. Pounds of quicksilver produced: 9,118.2.

Labor, man-hours per ton:	
Screening and sorting	0.950
Furnace treatment	.210
Flotation	.373
Retorting	.047
Condensing	.477
Assaying	.125
Miscellaneous	.176
Total	2.358
Tons per man per 8-hour shift	3.393
Power, kw. h. per ton:	
Screening and sorting	10.80
Furnace treatment	1.44
Flotation	2.32
Retorting	1.28
Assaying	0.64
Condensing	2.72
Miscellaneous	13.92
Total	33.12
Supplies:	
Oil consumption, gallons per ton of dry ore:	
Furnace treatment	22.8
Retorting	1.2
Total	24.0
Reagents, pounds per ton of ore:	
Sodium aerofloat	0.0543
Pine oil	0.3093
Barrett No. 4	0.0186
Lime	1.954
Labor, percentage of total cost	36.9
Power and supplies, percentage of total cost	63.1

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Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MILLING METHODS AND COSTS AT THE LEAD-ZINC CONCENTRATOR OF THE TREADWELL YUKON CO., LTD., AT TYBO, NEV.

BY

W. H. Blackburn

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

MILLING METHODS AND COSTS AT THE CONCENTRATOR OF THE TREADWELL YUKON CO., LTD., AT TYBO, NEV.[1]

By W. H. Blackburn[2]

INTRODUCTION

This paper describing the milling practice at the Tybo concentrator is one of a series of similar papers being prepared by the United States Bureau of Mines on milling methods and costs at the various mills in the United States.

ACKNOWLEDGMENT

A large part of this paper was prepared by H. M. Lewers, mill superintendent of the Tybo plant of the Treadwell Yukon Co., Ltd.

LOCATION

The Tybo mine and concentrator are situated in Nye County, Nev., 70 miles northeast of Tonopah, which is the nearest railroad station and general supply center. The district is accessible by automobile stage either from Tonopah or Ely over an improved gravel highway connecting the two towns. The climate is semiarid, and operations are conducted the year around without difficulty. The elevation at the collar of the shaft is 6,820 feet above sea level.

GENERAL

The concentrator is built on a hillside adjacent to the main hoisting shaft of the mine and was designed and constructed for an all-flotation flow sheet. Two classes of concentrates are produced; lead concentrates which average 90.43 ounces of silver per ton, 62.75 per cent of lead, and 3.50 per cent of zinc; and zinc concentrates which average 13.65 ounces of silver per ton, 2.71 per cent of lead, and 46.57 per cent of zinc. The lead concentrates are shipped to the American Smelting and Refining Co.'s reduction plant at Selby, Calif., and the zinc concentrates to the Sullivan electrolytic zinc plant at Kellogg, Idaho.

All supplies for the mine and mill are hauled from Tonopah in 10-ton capacity Fageol trucks, each equipped with a trailer. The company maintains a rail siding, warehouse, and loading facilities at Tonopah, from which place the concentrates are shipped. The combined capacity of a truck and trailer is about 20 tons of concentrates. A round trip between Tybo and Tonopah requires about 13 hours.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6430."

2 - One of the consulting engineers, U. S. Bureau of Mines, and manager, Nevada Mines, Treadwell Yukon Co., Ltd., Tybo, Nev.

The mill water is obtained from the mine, and as the supply is more than sufficient to meet the concentrator requirements, mill water is not reclaimed. The domestic water supply is obtained from a small creek above the camp.

Power is purchased from the Nevada-California Power Co. The Treadwell Yukon Co. constructed a branch line from Manhattan to Tybo, a distance of 40 miles, at a cost of about $1,000 per mile. Power is transmitted to Tybo at 55,000 volts, and is stepped down to 440 volts for mine and mill use.

ORE TREATED

The ores treated are complex in character, containing lead, zinc, iron, silver, and gold. The Tybo mine from which all the ores for the concentrator are derived, has been opened to a depth of 976 feet by a vertical three-compartment shaft. The principal vein has an average width of 5 feet, a maximum width of 30 feet, and an average dip of 76°. Mining is done by the shrinkage-stope method without artificial support, except in a few places where stulls are necessary to support a weak hanging wall, and by the stope-and-fill method where walls are soft.

A chemical analysis of a composite sample of the ores between the 710 and 400 foot levels of the mine follows.

Chemical analysis of composite sample of ore between the 710 and 400 foot levels

	Per cent	Ounces per ton
Silica	32.05	-
Iron	15.08	-
Alumina	11.48	-
Calcium oxide	7.73	-
Magnesium oxide	1.62	-
Phosphorous	0.24	-
Sulphur	16.56	-
Lead	7.38	-
Zinc	5.56	-
Copper	0.02	-
Nickel	Trace	-
Cobalt	None	-
Vanadium	None	-
Manganese	0.26	-
Barium sulphate	None	-
Arsenic	0.59	-
Antimony	0.13	-
Cadmium	0.06	-
Gold	-	0.02
Silver	-	10.81

Although most of the silver content in Tybo ore is associated with the galena, the sphalerite and pyrite are both argentiferous, the relative content in the last two minerals being greater in the upper-level ores than in the lower-level ores. Typical assays of pure minerals give the following results:

	Ounces per ton
Galena	110.0 to 130.0
Sphalerite (upper levels)	12.0 to 20.0
Sphalerite (lower levels)	7.0 to 10.0
Pyrite (upper levels)	8.0 to 11.0
Pyrite (lower levels)	3.0 to 6.0

Figure 1 shows a vertical section of the Tybo vein and gives the approximate degrees of oxidation in the ore body and the varying amounts of lead and silver associated with the pyrite and sphalerite in the different horizons.

Some oxidation of the ore is observed on the 710-foot level of the mine and this oxidation gradually increases towards the surface. Above the 400-foot level the ores have undergone considerable oxidation. While practically all of the completely oxidized ores were extracted in former mining operations confined to the area above the 400-foot level, large bodies of partly oxidized ores were left. These partly oxidized ores have comprised a considerable proportion of the concentrator feed up to the present time. The average of seven typical determinations on ores above the 400-foot level is as follows:

	Per cent
Lead in sulphide form	4.4
Lead in oxidized form	3.3
Total lead	7.7

An excessive amount of primary slimes, associated mainly with the partly oxidized ores from the upper levels of the mine, has caused considerable difficulty in concentrator treatment. These slimes are colloidal in character and if present in the mill feed above a critical amount have a tendency to upset completely the equilibrium of flotation operations and to cause lower-grade concentrates and lower recoveries. A large amount of slime in the flotation circuit produces excessive frothing, and when this takes place slime can not be prevented from floating with the concentrates, thus lowering the grade of both lead and zinc concentrates from 5 to 10 per cent in lead and zinc respectively. To overcome this difficulty the ores from the different levels of the mine are mixed as much as is practicable.

The relative distribution of metals in the coarse and fine sizes of mine ore is illustrated by the following analyses of products obtained by screening a sample of ore taken at the shaft bin through a 200-mesh sieve. The minus 200-mesh material amounted to 10.4 per cent of the total weight.

Analyses of ore samples taken at shaft bin and screened through a 200-mesh sieve

	Weight, per cent	Silver, ounces per ton	Lead, per cent	Zinc, per cent	Iron, per cent
Plus 200-mesh	89.6	11.2	7.1	5.0	13.1
Minus 200-mesh	10.4	13.0	15.2	5.9	7.7

The sphalerite so far treated is high in chemically combined iron, pure specimens containing from 50 to 55 per cent of zinc, and 10 to 15 per cent of iron, the iron replacing the zinc isomorphously in accordance with the formula (ZnFe)S. (Zinc sulphide containing 10 per cent or more iron is usually classified as "marmatite").

BRIEF HISTORY OF CONCENTRATOR OPERATIONS

Ore was first discovered in the Tybo district in 1869. In 1872 the Tybo Consolidated Mines Co. erected a lead smelter, operating two furnaces with a combined capacity of 80 tons per day. Locally burned charcoal was used as fuel. In 1879 the smelter closed down. Soon afterwards a 20-stamp mill was erected, and the ores were treated for their silver content by the "Reece River Process" - that is, pan amalgamation preceded by a chloridizing roast. The records indicate a recovery of 78 to 81 per cent of the silver on ores which averaged from 25 to 30 ounces of silver per ton. This plant operated until 1888, when it was abandoned, due to the exhaustion of the oxidized ores which were amenable to this process.

In 1917, the Louisiana Consolidated Mining Co. acquired the properties and erected a flotation concentrator and a lead smelter. This venture was unsuccessful, due to the high iron and zinc contents of the ores. No method was available at that time for the separation of these minerals. This plant closed down in 1920.

In 1925 the properties were optioned to the Treadwell Yukon Co. After extensive exploration work the option was exercised. Construction of the concentrator was started in November, 1928, and the mill began to produce on May 13, 1929.

PRESENT CONCENTRATOR METHODS

The general flow sheet of the concentrator is shown in Figure 2. The method of flotation employed is a selective separation of the lead and zinc minerals in the original pulp.

Crushing

All ore delivered to the primary ore bin has passed through grizzlies set at 9 inches which are located on the level stations of the mine over the ore pockets. Ore from the mine is delivered through the main vertical three-compartment shaft to the primary ore bin at the shaft collar. This bin has an inclined bottom, is steel lined, and has a capacity of 125 tons.

From the primary bin the ore is fed to a bar grizzly with 4-inch openings, set at an angle of 45°, by a 36-inch Link-Belt apron feeder. The grizzly oversize product goes to an 18 by 30 inch Allis-Chalmers Blake-type crusher set at 4 inches. The crusher is belt-driven at a speed of 250 r.p.m. by a 50-hp. motor. The average power required to operate the crusher under full load amounts to 16-1/2-hp. The crusher is equipped with manganese steel wearing plates having vertical corrugations, and during 12 months of operation the only replacement has been one stationary wearing plate. The normal crushing rate is 65 tons per hour.

The grizzly undersize joins the crusher discharge on a 24-inch, rubber-surfaced belt conveyor and is carried to a 3 by 8 foot Link-Belt vibrating screen, inclined at an angle of 18° and operated at a speed of 1,300 r.p.m. The screen is of manganese steel plate one-fourth inch thick, punched with 1-1/4-inch square holes, and lasts from three to four months. Due to the muddy and sticky character of the ore only a small amount of the undersize is larger than 3/8-inch size.

Surface

100 ft. — Lead, zinc and iron almost completely oxidized. Ore almost completely extracted in earlier operations. (1865 to 1888)

200 ft.

---

300 ft. — Lead, zinc, and iron from 30 to 50 per cent oxidized. Part of the oxidized ores extracted in earlier operations.

	Silver, ounces	Lead, per cent
Pyrite ------	12 to 15	3.5 to 5.2
Sphalerite -----	18 to 30	2.5 to 4.0

---

400 ft. — Lead, zinc and iron from 15 to 25 per cent oxidized.

	Silver, ounces	Lead, per cent
Pyrite ------	8 to 12	1.5 to 3.0
Sphalerite -----	14 to 18	2.0 to 3.0

500 ft.

Lead, zinc, and iron from 5 to 15 per cent oxidized.

	Silver, ounces	Lead, per cent
Pyrite ------	4 to 8	0.5 to 1.5
Sphalerite -----	6 to 10	0.8 to 2.0

600 ft.

Lead, zinc, and iron from 0 to 10 per cent oxidized.

	Silver, ounces	Lead, per cent
Pyrite ------	3.0 to 5.0	0.3 to 0.5
Sphalerite -----	5.0 to 8.0	0.7 to 1.0

700 ft.

Zone of partial oxidation and sulphide enrichment

Figure 1.--Vertical section of the Tybo vein, showing approximate degrees of oxidation of the ore body, and varying amounts of lead and silver associated with the pyrite and sphalerite in the different horizons

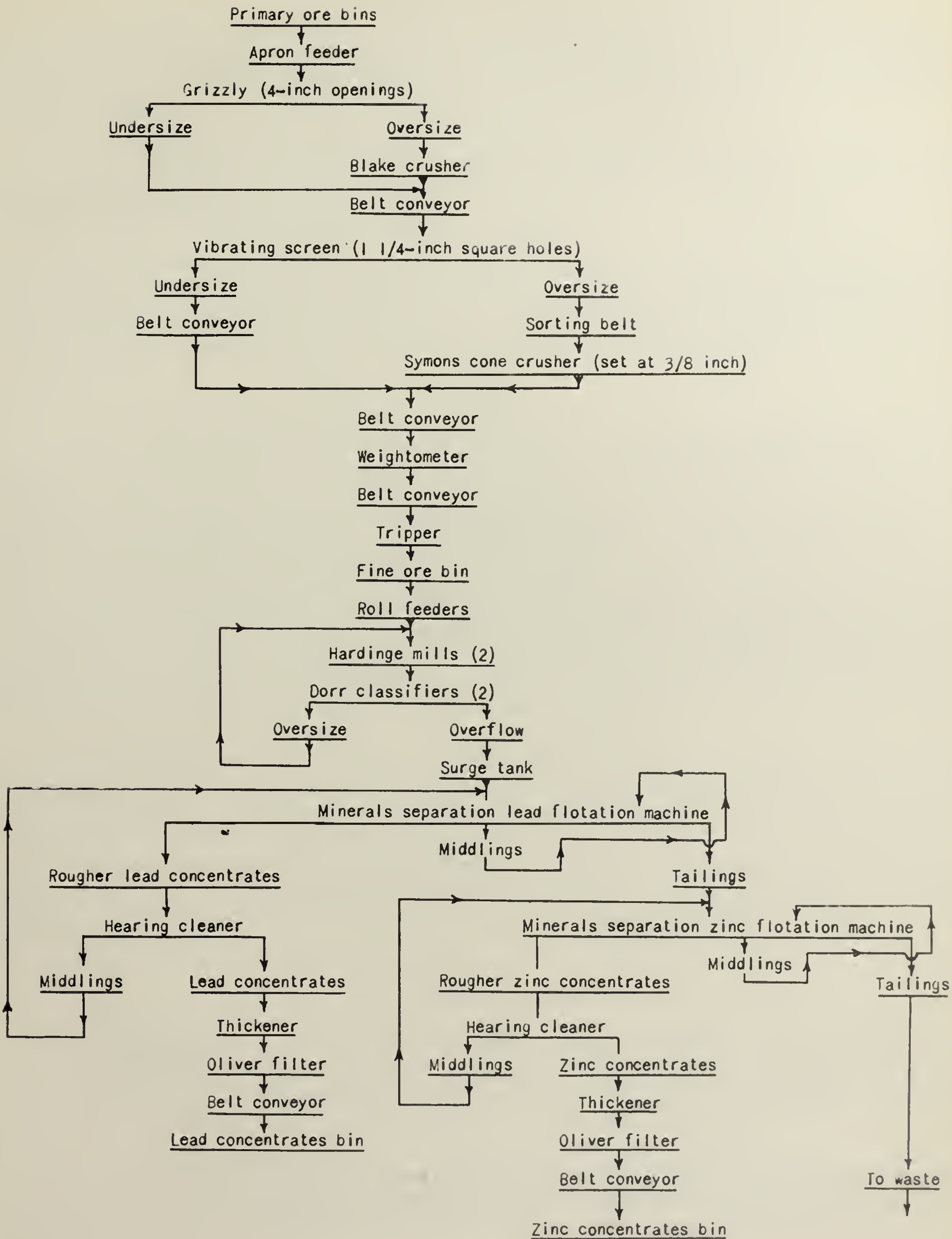

Figure 2.- Flow sheet of Tybo concentrator

The screen oversize goes to a 30-inch, rubber-surfaced belt conveyor where wood, tramp iron, and occasional pieces of waste are sorted out by hand. The ore is usually wet and covered with slime to such extent that it is difficult to distinguish waste from ore. To obtain efficiency from sorting, the ore would have to be thoroughly washed.

The oversize from the vibrating screen is discharged into a 4-foot Symons cone crusher set at 3/8-inch. The Symons crusher is direct-connected to a 100-hp. motor and consumes an average of 36-hp. under full load.

The undersize from the vibrating screen is conveyed on a 20-inch belt to a second 20-inch belt where it joins the discharge from the cone crusher. The combined products pass over a Merrick weightometer to a 20-inch distributing belt conveyor, equipped with a self-propelled tripper, which discharges the products into a 1,400-ton fine-ore bin.

Grinding

The fine-grinding equipment consists of two 8 by 4 foot Hardinge ball mills each operating in closed circuit with a type D, 6 by 25 foot, Dorr classifier. Each ball mill is driven at a speed of 19 r.p.m. by a 200-hp. motor, through a Farrell reduction unit, the reduction ratio being 870 to 159. The average power consumption, under full load, amounts to 139 hp. The ball mills operate with a ball load of 18 tons, a circulating load of approximately 400 per cent and a pulp density between 75 and 78 per cent of solids. Ball consumption, amounting to 2.7 pounds per ton of ore ground, is compensated by the addition of 3-inch "Adamantine" steel balls. Manganese steel liners are used, and their estimated average life is from 14 to 16 months. The feed-cone liners were replaced after 8 months of service, the original cone-discharge liners are still in service, and approximately 80 per cent of the original breast liners are in service after 13 months of operation.

Each Dorr classifier is operated at 14 strokes per minute by a 5-hp. motor which consumes an average of 1-1/2 hp. under full load. The overflow is maintained at a density of 32 to 36 per cent of solids, and these solids average from 68 to 75 per cent minus 200-mesh material. Table 1 gives screen analyses of the ball-mill discharge and classifier overflows. As previously stated, a large amount of slime is detrimental to flotation operations and since an excessive amount of slime is produced in grinding ores from the upper levels of the mine, finer grinding than is shown by the screen analyses of Table 1 is not practiced.

Lead Flotation Circuit

The classifier overflow products, which comprise the feed to the lead flotation circuit, flow by gravity to a surge tank, 24 feet in diameter by 8 feet high, constructed of redwood staves. The pulp flows by gravity from this tank to the head of the lead flotation section.

The equipment in the lead flotation section consists of one 16-cell Minerals Separation Sub-A machine and one three-pan Bunker Hill-Sullivan type Hearing pneumatic cleaner. The cells of the Minerals Separation machine are 18 by 31-1/4 inches and the impellers are 18 inches in diameter.

Depending on the grade and tonnage of the ore treated, the first three to five cells of the Minerals Separation machine produce rough flotation concentrates which are

pumped to the Hearing cleaner unit. The cleaner unit produces finished concentrates and a middlings product which returns by gravity to the head of the Minerals Separation machine. The froths from rougher cells 6 to 10, inclusive, are returned to the head of the rougher machine, froths from cells 11 to 13 are returned to cell 8, and froths from cells 14 to 16 are returned to cell 11. The tailings from the lead section comprises the feed of the zinc flotation circuit.

The spindles of the Minerals Separation machine are driven in pairs at a speed of 330 r.p.m. by eight 7-1/2-hp., alternating-current, 440-volt, vertical motors operating at 1,155 r.p.m. Tex-rope drives are used and after operating over a year are in excellent condition. The power requirement for the impellers of the Minerals Separation machine amounts to 2.6-hp. for each of the 16 impellers when operating under full load. The impellers of the lead section machine have a life of approximately eight months.

Reagents are introduced into the flotation pulp at the ball mills and just ahead of the flotation circuit. The following kinds and amounts of reagents per ton of ore treated are added to the ball mills.

	Pounds
Soda ash	1.30
Sodium cyanide	0.35
Zinc sulphate	0.50

The kinds and amounts of reagents added ahead of the lead flotation circuit per ton of ore treated follow:

	Pounds
Soda ash	0.60
Sodium cyanide	0.30
Zinc sulphate	0.40
Ethyl xanthate	0.10
Cresylic acid	0.50

Soda ash is added to neutralize the acidity of the ore and to maintain the desired alkalinity of the pulp. The alkalinity is determined by pH indicators and is maintained at from 8 to 8.6 pH values. Sodium cyanide and zinc sulphate are used in conjunction to depress the sphalerite and pyrite in the lead circuit. The ethyl xanthate is added as an accelerator in the flotation of lead minerals and the cresylic acid is used as a frothing agent.

The density of pulp in the lead circuit is maintained between 30 and 35 per cent of solids.

The character of the ore delivered to the concentrator varies considerably from day to day. Operating results from the treatment of clean sulphide ores derived from the lower levels of the mine are consistent and concentrates ranging from 66 to 70 per cent of lead are readily obtained. Variations from this condition are due to the degree of oxidation and amount of primary slime encountered in the concentrator heads. As the slime content of the flotation feed increases, the grade of concentrates produced decreases, and experience has shown that this condition is not to be overcome by the use and control of reagents.

Zinc Flotation Circuit

The tailings from the lead flotation section flow by gravity through an inverted syphon to the head of the Minerals Separation machine of the zinc circuit. The flotation equipment of the zinc section is a duplicate of that used for the flotation of lead minerals. Depending upon the grade and tonnage of the feed to the zinc section, rough concentrates are produced from the first four to eight cells. These concentrates are pumped to the Hearing cleaner unit, which produces finished concentrates and a middlings product which is returned by gravity to the head of the Minerals Separation rougher machine. Assuming that cells 1 to 4, inclusive, of the Minerals Separation machine are producing rough concentrates, the froths of cells 5 to 8, inclusive, are returned to the head of the rougher machine, and froths from cells 9 to 16, inclusive, are pumped to cell 5. The tailings of the zinc section are conveyed by launder to the tailings pond.

The following kinds and amounts of reagents per ton of original ore treated are added to the pulp ahead of the zinc section.

	Pounds
Copper sulphate	1.30
Ethyl xanthate	0.10
Hydrated lime	1.90
Pine oil	0.01

The copper sulphate is added to reactivate the sphalerite which was depressed in the lead circuit. Lime, which can not be used in the lead circuit on account of the action of the calcium ion in depressing galena, is used in the zinc circuit as a depressant of pyrite. Pine oil is used as an auxiliary frothing reagent.

The density of the pulp in the zinc circuit is maintained at approximately 25 per cent of solids. Due to the lower pulp density in the zinc circuit as compared to the pulp of the lead circuit, flotation-machine impellers of the zinc circuit have nearly twice the life of impellers operated in the lead circuit.

Power requirements for the impellers of the Minerals Separation machine of the zinc circuit are 2.4 hp. for each of the 16 impellers operating under full load.

Air is furnished to the Minerals Separation machines of both lead and zinc circuits at 1-1/2 pounds pressure by a General Electric centrifugal air compressor direct-connected to a 11-hp. motor consuming 8 hp. under full load. Air is furnished to the Hearing cleaner units of both lead and zinc sections at 4-1/2 pounds pressure by a No. 2 Connersville blower, belt-driven by a 10-hp. motor requiring 11 hp. under full load.

DEWATERING OF FLOTATION CONCENTRATES

The finished concentrates from the lead and zinc flotation circuits are pumped tc their respective Hardinge thickeners, each 24 feet in diameter by 8 feet high, by 2-inch Wilfley pumps. The scraping mechanism of the thickeners makes one revolution in six minutes. Thickener feeds contain about 10 per cent of solids and the thickener discharge from 60 to 70 per cent of solids. The thickened concentrates are conveyed through 4-inch pipe lines equipped with Nordstrom valves, to two Oliver filters, each 5 feet 4 inches in diameter by 10 feet wide. Each filter has 167-1/2 square feet of filtering surface, Palma twill style 15 C

being used as the filtering medium. The average life of the filtering cloth is eight months on the zinc filter and three months on the lead filter.

Vacuum at the filters is maintained at about 18 inches of mercury, the elevation of the plant being 6,700 feet above sea level. Two Oliver vacuum pumps, 14 by 8 inches in size, driven by two 15-hp. motors through Lenox short-center drives, are used for this purpose. Compressed air for removing the filter cake is supplied at 10 pounds pressure by one 9-1/2 by 8 inch Oliver compressor driven through a Lenox drive by a 10-hp. motor which requires 9-1/2 hp., on the average, under full load.

The filter cakes drop onto 16-inch belt conveyors and are discharged into their respective bins. The concentrates are transported by truck to Tonopah for rail shipment. No provision has been made for heating the concentrates storage bins in winter, and no trouble has been experienced with frozen concentrates, although the minimum temperature during winter reaches 10° below zero.

Table 2 gives analyses of lead and zinc concentrates.

DISPOSAL OF TAILINGS

The tailings are conveyed in launders by gravity to the disposal ground. Adequate ground is available for tailings storage without impounding, and their disposal does not constitute a serious problem.

SAMPLING AND CONTROL OF OPERATIONS

The flotation feed and the final flotation tailings streams are cut at 15-minute intervals by Galigher automatic samplers. Samples of the tailings of the lead section and of the two final concentrates are cut from the respective launders by operators. A specially designed hand sampler, shown in Figure 3, is used for this purpose.

The daily mill feed and tailings samples are composited for a 15-day period on the basis of tonnage treated, allowing 1 gram for each 10 tons of ore concentrated, and the compcsite sample is assayed by control methods.

A comparison of the average smelter returns with concentrator assays made on samples taken by hand from May 13 to December 31, 1929, follows:

Average smelter returns compared with concentrator assays of samples taken by hand from May 13 to December 31, 1929

	Lead concentrates			Zinc concentrates		
	Silver, ounces per ton	Lead, per cent	Zinc, per cent	Silver, ounces per ton	Lead, per cent	Zinc, per cent
Smelter ...	90.43	62.75	3.50	13.65	2.71	46.57
Concentrator	90.65	62.47	3.77	13.39	2.29	46.84

The production of lead concentrates is calculated from the usual formula:

Per cent of mill feed produced as lead concentrates = 100 F-T ÷ C-T
where F, C, and T are lead assays of lead section feed, lead section concentrates, and lead section tailings products, respectively.

The production of zinc concentrates is calculated from the formula:

Percentage of original mill feed produced as zinc concentrates = (100-L) (F-T) ÷ (C-T)
where F, C, and T are the zinc assays of zinc section feed, zinc section concentrates, and zinc section tailings products, respectively, and L is the per cent of mill feed produced as lead concentrates.

Flotation operations are controlled by the appearance of the froths which are frequently examined by panning with a white enameled vanning plaque.

The pressure filter shown in Figure 4 is used for the dewatering of pulp samples. This filter was designed by H. M. Lewers, mill superintendent, and has been found satisfactory. In operating the filter, a 7-inch diameter filter paper is placed over the canvas and 90-pound pressure mine air is used. It requires less than three minutes to dewater a pulp sample weighing 10 ounces and the filtrate is perfectly clear.

CONCENTRATOR RECOVERIES AND LOSSES

Table 3 gives concentrator results for March, 1930. Table 4 shows the gross values per ton of the heads and of the lead concentrates and zinc concentrates. Table 5 gives extractions of silver, lead and zinc computed as indicated in the table. Table 5 also gives the "economic recoveries" of silver, lead, and zinc. "Economic recovery" of a metal at this plant is computed as the product of the gross value of the metal in the concentrator heads, in dollars, and the recovery of the metal, as concentrates, in per cent. Table 6 gives the metallurgical data for March, 1930.

Most of the silver loss in the tailings is due to silver associated with pyrite, as indicated by the following test. A sample of concentrator tailings, representing one week of operation, was sized by a 200-mesh sieve. The minus 200-mesh material was further separated into sand and slime products, the sand product consisting of approximately minus 200-mesh plus 500-mesh sizes. A separation of the sulphides and gangue contents of the sand product was made with bromoform of 2.82 specific gravity. The sulphides obtained were given a flash or magnetizing roast, and the pyrite was then separated with a magnet. The following tabulation of assay values in the several products obtained indicates the association of silver with the pyrite.

	Silver,	Per cent		
	ounces per ton	Lead	Zinc	Iron
Composite tailings sample	2.76	1.1	1.3	11.6
Slime product, minus 500-mesh sizes	2.30	0.4	0.6	7.7
Bromoform float sand	0.50	Trace	Trace	-
Pyrite	10.25	1.5	0.1	-

The following tabulation gives a comparison of analyses of concentrator heads and final products when treating two different mixtures of lower and upper level mine ores and illustrates the effect of increasing the proportion of upper-level ores on the grade of concentrates.

Comparison of analyses of concentrator heads and final products when treating two different mixtures of lower and upper level mine

	Concentrator feed mixtures, per cent							
	Lower levels = 80 Upper levels = 20				Lower levels = 50 Upper levels = 50			
	Assays				Assays			
	Silver, ounces per ton	Lead, per cent	Zinc, per cent	Iron, per cent	Silver, ounces per ton	Lead, per cent	Zinc, per cent	Iron, per cent
Heads	8.6	5.7	5.0	11.1	9.1	5.8	4.3	12.5
Lead concentrates	96.4	69.5	4.3	6.1	82.8	63.2	4.2	6.7
Zinc concentrates	12.4	1.6	48.6	11.7	13.3	2.4	47.9	11.0
Tailings	2.0	0.87	1.03	12.9	3.0	1.6	0.95	13.2

The lead losses in the tailings produced during the periods covered by the preceding tabulation, were determined to be distributed as follows:

Distribution of lead losses in the tailings

	Concentrator feed mixtures, per cent	
	Lower levels = 80 Upper levels = 20	Lower levels = 50 Upper levels = 50
Lead in tailings, in oxidized form, per cent	0.35	0.80
Lead in tailings, associated with pyrite, per cent	.32	.60
Lead in tailings, association not determined, per cent	.20	.20
Total lead in tailings, per cent	0.87	1.60

The following tabulation gives a screen-assay analysis of the tailings product produced during the period in which the 50 per cent upper-level and 50 per cent lower-level ore mixture was treated.

Screen-assay analysis of concentrator tailings

Screen sizes	Weight, per cent	Assays				Per cent of total			
		Silver, ounces per ton	Lead, per cent	Zinc, per cent	Iron, per cent	Silver	Lead	Zinc	Iron
Plus 100-mesh	19.0	2.2	0.8	0.9	5.1	15.2	9.4	18.1	8.7
Minus 100 plus 150 mesh	13.9	1.6	1.4	1.1	15.5	8.0	11.9	16.2	19.4
Minus 150 plus approximately 350 mesh sands	20.9	1.8	2.0	0.8	19.2	13.7	25.6	17.7	36.1
Minus approximately 350 plus 500 mesh sands	4.3	3.8	0.6	0.7	9.6	5.7	1.9	3.2	3.7
Slimes	41.9	3.8	1.9	1.0	8.5	57.4	51.2	44.8	32.1

CONCENTRATOR OPERATION AND COSTS

Under normal conditions of treating 320 tons of ore per day the mill crew is distributed as follows:

	Number of operators
Crusher	2
Sorting belt	1
Ball mills	3
Flotation	3
Filter and reagents	1
Total	10

The entire concentrator is shut down for a short period twice each month for the examination of ball-mill liners and for general repairs. Repairs and oiling are handled by the mechanical and electrical departments. The total time lost because of enforced shut downs in over a years operation amounted to two hours, caused by defective insulation in a small motor.

The milling costs vary inversely with the tonnage of ore treated. A summary of costs per ton of original ore, for March, 1930, when treating 320 tons of ore per day, is given in Table 7. Distributions of labor and power and a summary of reagents are shown in Table 8.

Table 1.--Screen analyses of concentrator products

Screen sizes, mesh	Ball-mill discharges, 78 per cent solids		Classifier overflows, 42 per cent solids		Flotation tailings of lead section, 33 per cent solids		Flotation tailings of zinc section, 25 per cent solids	
	Weight, per cent	Cumulative per cent	Weight, per cent	Cumulative per cent	Weight, per cent	Cumulative per cent	Weight, per cent	Cumulative per cent
On 14	9.75	9.75	-	-	-	-	-	-
On 28	4.15	13.90	-	-	-	-	-	-
On 35	3.90	17.80	-	-	-	-	-	-
On 48	5.91	23.71	-	-	-	-	-	-
On 65	10.30	34.01	5.09	5.09	6.12	6.12	6.44	6.44
On 100	11.35	45.36	9.10	14.19	9.36	15.48	8.07	14.51
On 150	10.10	55.46	8.86	23.05	9.15	24.63	10.92	25.43
On 200	6.77	62.23	8.10	31.15	8.41	33.04	7.10	32.53
Through 200	37.77	100.00	68.85	100.00	66.54	99.58	67.47	100.00

Table 2.--Analyses of lead and zinc concentrates

	Ounces per ton		Per cent								
	Gold	Silver	Lead	Zinc	Copper	Iron	Sulphur	Insoluble	Arsenic	Antimony	Cadmium
Lead concentrates	0.18	90.43	62.75	3.70	Trace	6.7	18.3	3.8	1.3	0.7	(1)
Zinc concentrates	0.04	13.65	2.71	46.57	0.10	11.1	28.4	3.3	(1)	(1)	0.5

1 - Not determined.

Table 3.--Concentrator results for March, 1930

	Weight		Assays			Weight, pounds per ton		Total weight			Concentration ratios
	Per cent	Tons	Silver, ounces per ton	Lead, per cent	Zinc, per cent	Lead	Zinc	Silver ounces	Lead, pounds	Zinc, pounds	
Concentrator heads	100.00	9 930.0	9.44	5.95	4.14	119.0	82.8	93,739	1,181,670	822,204	-
Lead concentrates	7.08	703.5	87.10	59.90	4.00	1,198.0	80.0	61,275	842,793	56,280	14.12 to 1
Zinc concentrates	6.29	625.0	14.28	2.55	47.60	51.0	952.0	8,925	31,875	595,000	15.98 to 1
Lead section tailings	92.92	9,226.5	3.60	1.85	4.17	37.0	83.4	33,215	341,381	769,490	-
Zinc section tailings	86.63	8,601.5	3.02	1.85	1.01	37.0	20.2	25,976	318,256	173,750	-

Table 4.--Gross metal values per ton of heads and of lead and zinc concentrates

	Metal prices	Heads	Lead concentrates	Zinc concentrates
Silver	41 cents per ounce	$3.87	$35.71	$5.85
Lead	5.50 cents per pound	6.54	65.89	[1]2.55
Zinc	5.00 cents per pound	4.14	-	47.60
Total		$14.55	$101.60	$56.00

1 - Price of lead in zinc concentrates taken at 5 cents per pound.

Table 5.—Per cent recoveries and "economic recoveries" of metals

	Silver	Lead	Zinc
Extractions:			
Extractions computed from metals in heads and concentrates, per cent	74.9	74.0	72.4
Extractions computed from metals in heads and tailings, per cent	72.3	73.1	72.0
Extractions computed from metals in concentrates and tailings, per cent	73.0	73.3	72.1
"Economic recoveries":			
Silver = 74.9 per cent of $3.87	$2.90	-	-
Lead = 74.0 per cent of $6.54	-	$4.84	-
Zinc = 72.4 per cent of $4.14	-	-	$3.00
Average "economic recovery" = 10.47/14.55 x 100 per cent		73.8	

Table 6.—Metallurgical data for March, 1930

Dry ore treated, tons	9,930
Moisture in ore to mill, per cent	3.8
Hours operated per day	24
Days operated	31
Ore treated per 24 hours, tons	320
Total concentrates produced, dry tons	1,328.5
Average lead concentrates produced per 24 hours, dry tons	22.7
Average zinc concentrates produced per 24 hours, dry tons	20.2
Recovery of lead, per cent	74.0
Per cent of total lead in lead concentrates	71.3
Per cent of total lead in zinc concentrates	2.7
Recovery of zinc, per cent	79.2
Per cent of total zinc in zinc concentrates	72.4
Per cent of total zinc in lead concentrates	6.8
Recovery of silver, per cent	74.9
Per cent of total silver in lead concentrates	65.4
Per cent of total silver in zinc concentrates	9.5
Ratio of concentration for lead section, tons of original ore into 1	14.12
Ratio of concentration for zinc section, tons of original ore into 1	15.89
Water consumption per ton of ore, tons	5
Ball consumption per ton of ore, pounds	2.7
Average pulp density in lead circuit, per cent of solids	34
Average pulp density in zinc circuit, per cent of solids	25
Average temperature of mill water, °F.	58

Table 7.—Summary of concentrator costs for March, 1930

Ore treated = 9,930.0 tons

Lead concentrates produced = 703.5 tons

Zinc concentrates produced = 625.0 tons

	Operating labor	Power	Supplies	Miscellaneous	Total
Sorting	$0.017	–	–	$0.001	$0.018
Crushing	.035	$0.023	$0.019	.002	.079
Grinding and classification	.054	.210	.176	.005	.445
Flotation	.073	.064	.359	.006	.502
Filtering	.015	.005	–	.001	.021
General mill expense	.050	.016	–	.151	.217
Total	$0.244	$0.318	$0.554	$0.166	$1.282

Table 8.—Distributions of labor and power and summary of reagents for March, 1930

Ore treated = 9,930.0 tons

Lead concentrates produced = 703.5 tons

Zinc concentrates produced = 625.0 tons

Labor (man-hours per ton concentrated):	
Sorting	0.027
Crushing	.047
Grinding and classification	.078
Flotation	.097
Filtering and mixing reagents	.022
Miscellaneous	.067
Total man-hours per ton of ore concentrated	0.338
Power, kw.h. per ton:	
Crushing	1.77
Grinding and classification	15.84
Flotation	4.85
Filtering	0.39
Miscellaneous	1.19
Total kw.h. per ton of ore concentrated	24.04
Reagents (pounds per ton of ore treated):	
Soda ash	1.90
Sodium cyanide	0.65
Zinc sulphate	0.90
Copper sulphate	1.30
Ethyl xanthate	0.20
Cresylic acid	0.50
Lime (hydrated)	1.90
Pine oil	0.01
Miscellaneous:	
Steel balls, pounds per ton of ore concentrated	2.70

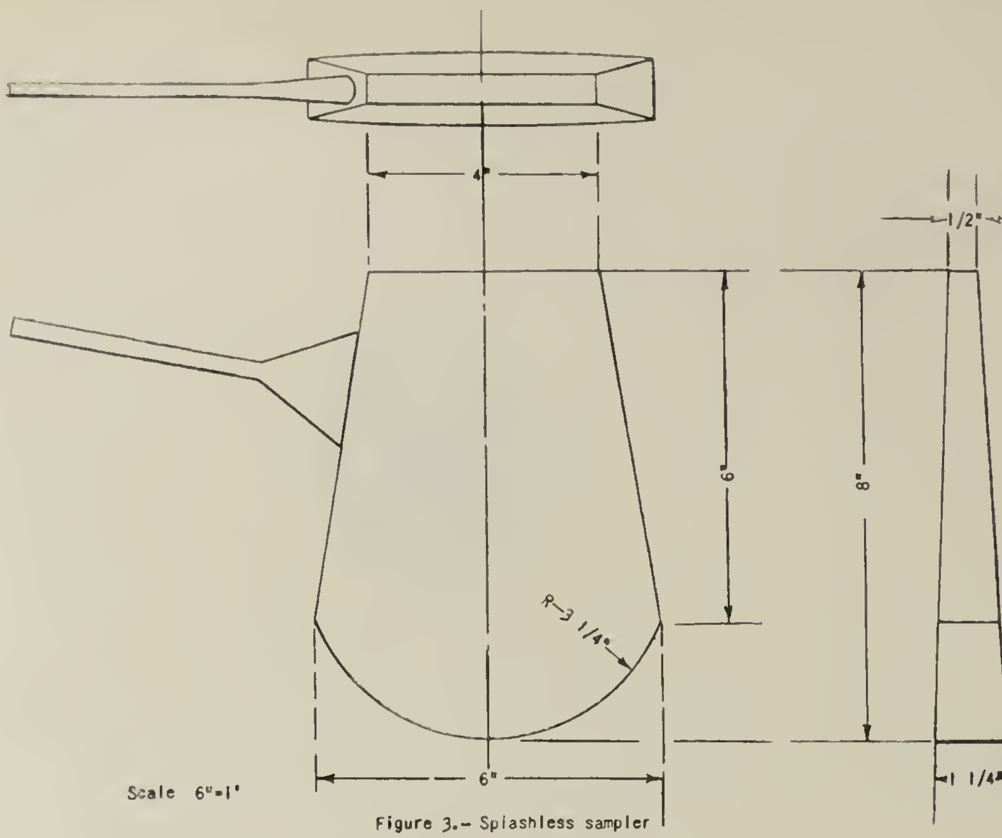

Figure 3.- Splashless sampler

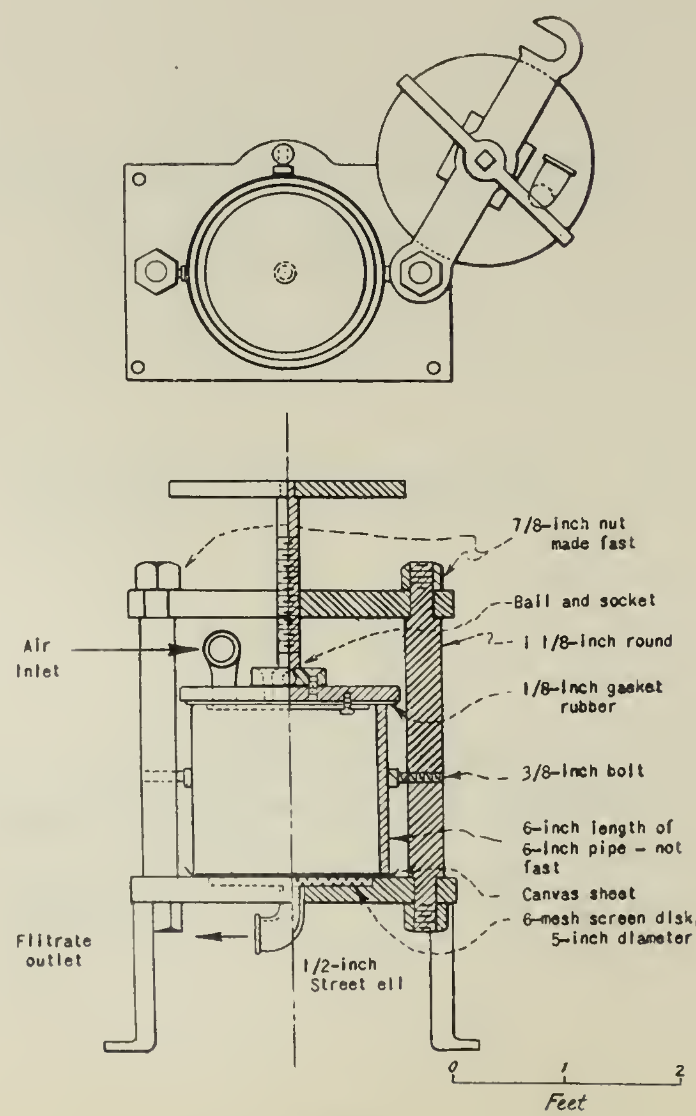

Figure 4.- Laboratory filter

I. C. 6431

May, 1931

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

STREET PAVING IN REPRESENTATIVE AMERICAN CITIES, 1925-1929

BY

Arthur H. Redfield

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May, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

STREET PAVING IN REPRESENTATIVE AMERICAN CITIES, 1925-1929[1/]

By Arthur H. Redfield[2/]

TABLE OF CONTENTS

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6431."

2/ Associate economic analyst, petroleum economics division, U. S. Bureau of Mines.

INTRODUCTION

The expansion of city street paving which characterized the ten years ended December 31, 1929, according to a survey of 201 representative cities made by the Bureau of Mines, reached its climax in 1927, with decreases reported for 1928 and 1929. The upward trend, followed by a decline, was general in cities of all sizes included in this study, but was most pronounced in cities of 40,000 to 100,000 inhabitants and in the Northeastern, Southeastern, and Southwestern States. In the North-Central States the trend of paving was downward from 1925 to 1929; in the Pacific and Rocky Mountain States it was generally upward.

Over the nation as a whole, bituminous types of paving continued to lead, furnishing two-thirds of the total street pavement laid in the 201 cities from 1925 to 1929. The actual yardage of bituminous pavement laid in these cities, however, was 3.8 per cent less in 1929 than in 1925, and the ratio of bituminous pavement to the total pavement dropped from 70.4 per cent in 1925 to 62.1 per cent in 1928, rising to 65.1 per cent in 1929. The area paved with Portland-cement concrete amounted to only one-fourth of the total pavement laid in these cities from 1925 to 1929, but the yardage of Portland-cement pavement increased by half during the period, and its share in the total pavement grew from 19.3 per cent in 1925 to 30.0 per cent in 1928, declining to 28.5 per cent in 1929. Brick, block, and stone pavements, which constituted the remaining one-twelfth of the total pavement laid in the 201 cities, decreased one-third in yardage, and declined in relative importance from 10.3 per cent in 1925 to 6.6 per cent in 1929.

Local preferences, based on such various factors as availability and cost of the different paving materials, cost of initial laying, local traffic conditions, or even custom and tradition, largely govern the choice of the type of street pavement. For example, bituminous pavements constituted 93.3 per cent of the total street surfacing laid in Michigan from 1925 to 1929, inclusive, but only 52 per cent of the total pavement in the adjacent State of Ohio, 57.9 per cent in Indiana, and 56.9 per cent in Illinois. In New Jersey they constituted 90.3 per cent of the whole, but in New York only 79.1 per cent, and in Pennsylvania only 73.5 per cent. In Massachusetts they amounted to 82.0 per cent of the total pavement, and in Connecticut to 97.0 per cent. In like manner, Portland-cement concrete formed 40.9 per cent of the total pavement in Indiana, but only 33.2 per cent in Illinois, 17.0 per cent in Ohio, and 6.0 per cent in Michigan. In the Middle Atlantic States, Portland-cement concrete made up 10.4 per cent of the whole in Pennsylvania, but only 6.9 per cent in New York, and 5.0 per cent in New Jersey. In New England 7.5 per cent of the street area in Massachusetts was surfaced with Portland-cement concrete but only 2.7 per cent in Connecticut. Similarly, brick, block, and stone pavements formed 30.8 per cent of the whole area in Ohio, but only 1.2 per cent in Indiana, 0.7 per cent in Michigan, and 9.9 per cent in Illinois. In Pennsylvania they constituted 16.0 per cent of the entire street surface; in New York, 14.0 per cent; but in New Jersey, only 4.7 per cent.

Size of cities was more influential in determining the distribution of certain types of pavement than of others. Differences in paving practice between the larger and the smaller cities were more marked in the case of the bituminous pavements. Sheet asphalt was favored by the larger cities; asphaltic concrete in the medium-sized and smaller cities; and asphaltic macadam in the smaller cities of the group studied. The use of Portland-cement concrete in each group of cities according to size was generally proportional to the share of each group in the total paving. Most of the brick, block, and stone was laid in the largest cities of over 1,000,000 population and in the smallest cities, having 40,000 to 100,000 inhabitants.

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SCOPE AND METHOD OF INVESTIGATION

The foregoing conclusions were based on statistics, compiled by the Bureau of Mines in connection with the asphalt chapter of Mineral Resources of the United States, of street pavement laid in 201 representative cities of the United States from 1925 to 1929, inclusive. Since street paving and road surfacing constitute the largest single commercial use of asphalt, accounting for 44 per cent of all asphalts sold in the United States, trends in street and highway paving form a factor of economic importance in relation to the national demand for asphalt. Moreover, it is important to know whether increased or decreased sales of paving asphalt in any given year are in conformity with the general trend of paving and road building. Statistics of rural highway construction, both by States and by counties and townships, are compiled and published by the Bureau of Public Roads of the Department of Agriculture. No public agency, however, has hitherto undertaken to compile statistics of city street paving. It is the purpose of this study to supply in part that lack and to furnish a statistical basis for estimating trends in the laying of street pavements in general and of asphaltic pavements in particular.

Inquiries were addressed to the city engineers, superintendents of highways, or corresponding officials of 230 municipalities with populations exceeding 40,000 according to the estimates of the Bureau of the Census for 1928. Where no city or town in a given State contained as many as 40,000 inhabitants, inquiry was made of the largest city or town in that State. The officials in charge of street paving were requested to report the number of square yards of each type of street pavement laid in their respective cities for a series of years ended December 31,1929. Sufficient data for the purposes of the inquiry were received from 201 cities, distributed over 46 States and the District of Columbia and containing an aggregate population of 42,376,000 in 1930. The cities included in the survey accordingly comprise 94.5 per cent of the population of the United States living in communities of over 40,000 inhabitants. The

survey covers all of the population living in cities of over 1,000,000 inhabitants, and all living in the cities containing from 500,000 to 1,000,000 residents, as well as 96.7 per cent of the population residing in cities of 250,000 to 500,000 inhabitants, 93.0 per cent of the population living in cities of 100,000 to 250,000, and 80.3 per cent of the population enrolled in communities of 40,000 to 100,000.

No endeavor was made to include all communities in the present figures and no effort was made to account for all the asphalt, Portland cement, vitrified brick, stone block or other materials used for paving within the United States in any one year. The purpose of the survey was to obtain over a series of years comparable figures from a sufficient number of representative American cities to permit the study of trends or tendencies in city street paving. While the collection of figures from cities of less than 40,000 inhabitants would be of interest, consideration of the time and labor involved compelled the restriction of the investigation to such a number of cities as could be conveniently handled.

One of the difficulties of such a study is the lack of uniformity in the method of keeping statistics of street paving from city to city. For instance, in some cities paved alleys as well as gutters and curbs are included, but for most cities the figures refer solely to the central roadway of traveled streets. Statistics of new pavement and of resurfacing are carefully distinguished in some cities and are combined in others. Although most of the cities comprised in this survey kept their paving statistics by the calendar year, several important cities used fiscal years beginning on various dates. Figures for such cities have been placed in the calendar year to which each fiscal year most closely corresponds. Nevertheless, the statistics received from each city are on the same basis from year to year, and it is felt that a summation of such statistics will provide an accurate analysis of existing tendencies in street paving.

In general, both new pavement and resurfacing are included in the tables in this report. Maintenance, repair, and patching work have for the most part been excluded. In the first place, comparatively few cities keep adequate statistics of the yardage of pavement laid in maintenance or repair. In the second place, such figures reveal little or nothing about the tendencies of paving. To a large extent a roadway is maintained or repaired with the same material of which it is constructed, and the element of free choice of material does not enter to the same degree that it does into new construction or into resurfacing.

Such factors as density of population, amount of taxable property, volume and nature of traffic, availability and cost of the various paving materials, and supply and cost of labor differ from section to section, from State to State, and even from city to city. There are marked differences in the paving practices of the various sections of the United States and between the larger and the smaller cities. Moreover, a five-year period does not adequately reveal tendencies in the construction of street improvements designed to last a generation with proper care. Of necessity, activity in paving is cyclical, especially

in the smaller and less wealthy municipalities. More than one city, after raising money by a bond issue, has laid in a single year enough street pavement to meet its requirements for a decade, and in the next four or five years has limited its activities to maintenance and repair of existing pavements and to minor extensions of existing streets.

In considering trends, it is important to recognize that the various materials used for street-paving--asphalt, Portland cement, vitrified brick, stone block, and wood block--are not necessarily competitive; to a large extent they supplement each other. Sheet asphalt is used as a surface course over a base of Portland-cement concrete, vitrified brick, or stone block. Asphaltic concrete and sheet asphalt are used to patch and resurface worn pavements of Portland-cement concrete, vitrified brick, or stone. Liquid asphalt and asphaltic joint-filler are used to fill joints, crevices, and cracks in Portland-cement concrete, brick, or stone pavements. Portland-cement concrete and vitrified brick are used for gutters, curbs, and shoulders to bituminous roadways, as well as for bases for asphaltic surfaces. Portland-cement concrete has been used for patching and resurfacing cement, brick, and stone block pavements.

GENERAL DISCUSSION

The peak of street paving in the 201 cities was reached in 1927, when 57,944,251 square yards was laid. This was an increase of 8.2 per cent over the 53,538,760 square yards laid in 1926, and of 22.5 per cent over the 47,286,585 square yards in 1925. From the maximum attained in 1927, the 53,494,925 square yards laid in 1928 constituted a decrease of 7.7 per cent. Similarly, the 48,868,036 square yards laid in 1929 represented a decrease of 6.8 per cent from the yardage laid in 1928, but an increase of 3.3 per cent over that laid in 1925.

Two-thirds (65.4 per cent) of the total street area paved in the 201 cities from 1925 to 1929, inclusive, was surfaced with bituminous types of paving. These include petroleum asphalt or lake asphalt mixed with sand or crushed stone, in varying proportions; liquid or semiliquid asphalt or road oil applied to the surface of a macadam, gravel, or earth roadway; and native rock asphalt laid as a surface. A small and relatively insignificant yardage of coal-tar mixtures is included in the total bituminous pavement. One-fourth (25.3 per cent) of the total street area was surfaced with Portland-cement concrete. This does not include Portland-cement concrete used as a base for sheet asphalt pavements or for gutters, curbs, and shoulders to bituminous roadways. Only one-eleventh of the total street area was paved with vitrified brick, wood block, or stone block or slab.

Total area of street pavement laid in 201 cities, 1925-1929, by types of paving
(In square yards)

	1925	1926	1927	1928	1929
Sheet asphalt	18,776,011	19,489,373	21,404,872	19,135,125	17,932,095
Asphaltic concrete	9,845,142	12,188,700	9,759,280	9,729,686	8,228,577
Asphaltic macadam	1,346,935	1,502,708	1,369,836	1,719,319	2,586,820
Asphalt block	416,523	961,495	991,092	203,987	133,788
Natural rock asphalt ...	852,725	814,444	1,163,335	1,099,940	1,086,780
Other asphaltic types ..	1,800,582	1,519,812	1,615,689	1,295,470	1,699,876
Tar macadam	114,433	39,055	27,917	10,099	10,000
Total bituminous	33,152,351	36,515,587	36,332,021	33,193,626	31,677,936
Portland-cement concrete	9,050,054	11,407,851	15,908,957	15,889,144	13,777,081
Brick, block, and stone	5,084,180	5,615,322	5,703,273	4,412,155	3,413,019
Total	47,286,585	53,538,760	57,944,251	53,494,925	48,868,036

The distribution of street paving by sections of the country is by no means proportional to the population. The 113 cities which represent the Northeastern district contained 68.1 per cent of the total population of the 201 cities, but laid only 55.5 per cent of the total pavement put down in the United States from 1925 to 1929, inclusive. On the other hand, the 22 cities of the Pacific-Rocky Mountain district contained only 10.2 per cent of the total population but laid 20.5 per cent of the total pavement. Similarly, the 28 Southeastern cities contained 7.5 per cent of the total population but laid 9.1 per cent of the total pavement; and the 18 cities of the North-Central district contained 5.3 per cent of the total population and laid 6.3 per cent of the total pavement. Cities of the Southwestern district, containing 8.9 per cent of the total population, laid 8.6 per cent of the total pavement.

Total area of street pavement laid in 201 cities, 1925-1929,
by geographic districts
(In square yards)

	1925	1926	1927	1928	1929
Northeastern	26,711,257	28,740,042	32,379,220	29,711,606	27,288,789
Southeastern	4,727,107	5,696,882	5,956,072	3,714,401	3,621,420
Southwestern	4,036,221	4,427,700	4,942,712	4,509,577	4,609,738
North Central	3,853,858	3,471,029	3,574,668	2,792,526	2,755,630
Pacific-Rocky Mountain	7,958,142	11,203,107	11,091,579	12,766,815	10,592,459
Total	47,286,585	53,538,760	57,944,251	53,494,925	48,868,036

From 1925 to 1929 the cities and towns with populations exceeding 500,000 laid less than their proportionate share of the total paving. The five Class A cities with populations exceeding 1,000,000--New York, Chicago, Detroit, Philadelphia, and Los Angeles--contained 35.6 per cent of the total population comprised in this survey but laid only 32.1 per cent of the total street pavement.

The eight Class B cities of 500,000 to 1,000,000 inhabitants--Baltimore, Boston, Buffalo, Cleveland, Milwaukee, Pittsburgh, St. Louis, and San Francisco--containing 13.5 per cent of the total population comprised in this survey laid only 9.3 per cent of the total pavement. By 1925 the larger, wealthier, and longer settled municipalities had already a large proportion of their street area surfaced. The 24 Class C 3/ cities of 250,000 to 500,000 inhabitants, which contained 18.9 per cent of the total population, laid 16.3 per cent of the total pavement. The 51 Class D 4/ cities, with 100,000 to 250,000 residents, contained 16.0 per cent of the total population, and laid 16.5 per cent of the entire area of street surfacing. On the other hand, the 113 Class E 5/ municipalities, with populations of less than 100,000, contained only 16.0 per cent of the total population, but laid 25.7 per cent of the street pavement. These communities, especially those which have grown rapidly in the last five years, have made greater efforts to pave streets in the territories which they have annexed or into which they have expanded.

Total area of street pavement laid in 201 cities, 1925-1929, by population groups
(In square yards)

	1925	1926	1927	1928	1929
Class A ...	13,083,312	15,884,959	18,856,836	18,714,749	17,315,158
Class B ...	4,083,575	4,717,941	5,325,020	5,111,777	5,011,890
Class C ...	8,522,775	8,263,222	9,104,521	8,484,832	8,296,574
Class D ...	8,244,707	9,394,458	8,927,954	8,672,320	7,912,153
Class E ...	13,352,216	15,278,180	15,729,920	12,511,247	10,332,261
Total ...	47,286,585	53,538,760	57,944,251	53,494,925	48,868,036

Nevertheless the net increase of 3.3 per cent in the total pavement from 1925 to 1929 was due to the activities of the larger cities; in the smaller cities there was a net decrease in paving, especially in 1928 and 1929. Whereas the five Class A cities laid 32.3 per cent more street pavement in 1929 than in 1925, and the eight Class B cities 22.7 per cent more, the 24 Class C cities laid 2.6 per cent less pavement in 1929 than in 1925, the 51 Class D cities 4.0 per cent less, and the 113 Class E cities 22.6 per cent less.

DISCUSSION BY TYPES

Bituminous Pavements

The amount of bituminous surfacing used on city streets ran fairly parallel to the general trend for all types of paving but reached its peak in 1926 instead of 1927. After increasing from 33,152,351 square yards in 1925 to 36,515,587 square yards in 1926, the total yardage of bituminous pavements declined to 36,332,021 square yards in 1927, to 33,193,626 square yards in 1928,

3/ For list of cities included in Class C, see footnote 6, p.28.
4/ For list of cities included in Class D, see footnote 7, p.29.
5/ For list of cities included in Class E, see footnote 8, p.30.

and to 31,677,936 square yards in 1929. From 1927 to 1928, however, the decrease of 8.6 per cent in the area of bituminous paving coincided with a nation-wide decline of 7.7 per cent in all types of street paving from 1927 to 1928 but the decrease of 4.6 per cent from 1928 to 1929 in the area of bituminous paving laid fell short of the general decrease of 8.6 per cent in the yardage of all types of pavement. Bituminous types constituted 70.1 per cent of the total street pavement laid in 1925, 68.2 per cent in 1926, and 62.7 per cent in 1927. But, though the actual yardage of bituminous pavement laid decreased from 1928 to 1929, its ratio to the total pavement laid increased from 62.1 per cent in 1928 to 64.8 per cent in 1929.

Nearly four-sevenths (56.6 per cent) of all bituminous surfacing laid in the 201 cities from 1925 to 1929, inclusive, consisted of sheet asphalt. Nearly three-tenths (29.1 per cent) of the total comprised various types of asphaltic concrete. Included under this head are asphaltic concrete, "stone-filled sheet asphalt," warrenite-bitulithic, amiesite, Topeka, and willite. Only one-twentieth of the whole was formed by asphaltic penetration macadam. Native rock asphalt was used to surface only 2.9 per cent and asphalt block only 1.6 per cent of all street area paved with bituminous compounds. Miscellaneous bituminous types of surfacing, including national, bessonite, filbertine, oil macadam, coal-tar macadam, and others, furnished only 4.6 per cent of the whole.

Seven-tenths of all bituminous surfacing laid from 1925 to 1929, inclusive, in the 201 cities comprised in this survey was laid east of the Mississippi River. Three-fifths (61.9 per cent) of the total was put down in the Northeastern district, which lies north of the Potomac and Ohio Rivers and east of the Mississippi River and Lake Michigan. One-twelfth (8.9 per cent) of the total was laid in the Southeastern district, which occupies the area south of the Potomac and Ohio Rivers and east of the Mississippi and Pearl Rivers. Little more than one-eleventh (9.2 per cent) was used to surface city streets in the Southwestern district, lying west of the Mississippi and Pearl Rivers and south of St. Louis, Kansas City, Wichita, Amarillo, and El Paso. In the North-Central district, which extends west of Lake Michigan and the Mississippi River, north of St. Louis and Kansas City, and east of the Rocky Mountains, only 3.8 per cent of all the asphaltic pavements were laid. The remaining 16.2 per cent of all asphaltic pavements laid in the United States were put down in the Pacific-Rocky Mountain district, west of Great Falls, Cheyenne, Denver, Albuquerque, and El Paso.

Total area of bituminous street pavement laid in 201 cities, 1925-1929, by geographic districts
(In square yards)

	1925	1926	1927	1928	1929
Northeastern.........	20,230,775	21,573,507	22,534,567	21,386,441	20,008,024
Southeastern	3,006,718	3,302,104	4,181,856	2,297,148	2,460,448
Southwestern	2,901,661	3,098,945	3,497,737	3,173,950	2,981,378
North Central	1,818,340	1,490,024	1,335,433	1,026,948	894,121
Pacific-Rocky Mountain	5,194,857	7,051,007	4,782,428	5,309,139	5,333,965
Total	33,152,351	36,515,587	36,332,021	33,193,626	31,677,936

In both the Northeastern and the Southwestern districts the yardage of bituminous pavement laid each year followed the national trend of bituminous paving, rising to a peak in 1927 and falling off in 1928 and 1929. In the Southeastern district both the increase from 1925 to 1927 and the decrease in 1928 and 1929 were proportionately much larger. A marked decrease both in quantity and in proportion took place in the North-Central district where bituminous paving fell off at an almost steady rate, until the yardage laid in 1929 was only 49.2 per cent of that laid in 1925. In the Pacific-Rocky Mountain district bituminous paving increased 35.7 per cent to its peak in 1926, then dropped 32.2 per cent in 1927, and rose 11.0 per cent in 1928 to a level which it maintained almost unchanged in 1929.

In comparison with the size of cities, the distribution of asphaltic surfacing of all types has been roughly parallel to the distribution of street paving in general. The five Class A cities, which laid 30.0 per cent of all street pavement, put down 33.3 per cent of the total asphaltic pavement. The eight Class B cities laid 9.9 per cent of the total pavement and 9.2 per cent of the asphaltic pavement. The 24 Class C cities which laid 17.1 per cent of the total pavement put down 14.6 per cent of the asphaltic types. The 51 Class D cities, which laid 16.2 per cent of the total pavement, laid 17.1 per cent of the asphaltic pavements. The 113 Class E cities laid 25.7 per cent of the total pavement and 25.8 per cent of the bituminous surfacing.

Total area of bituminous street pavement laid in 201 cities, 1925-1929, by population groups
(In square yards)

	1925	1926	1927	1928	1929
Class A ...	9,670,459	11,457,158	11,941,319	11,760,930	12,007,878
Class B ...	2,721,134	3,184,898	3,330,687	3,271,010	3,198,191
Class C ...	5,745,185	5,170,176	4,885,034	4,442,249	4,618,732
Class D ...	5,621,238	6,005,263	5,845,914	5,504,763	4,745,362
Class E ...	9,394,235	10,698,092	10,329,067	8,214,674	7,107,773
Total .	33,152,351	36,515,587	36,332,021	33,193,626	31,677,936

At the same time the use of bituminous compounds to surface city streets increased in the larger cities, but decreased in the smaller cities, during the period under review. Although the five Class A cities laid 24.6 per cent more bituminous pavement in 1929 than in 1925 and the eight Class B cities 17.5 per cent more, the 24 Class C cities laid 19.6 per cent less bituminous pavement in 1929 than in 1925; the 51 Class D cities, 15.6 per cent less; and the 113 Class E cities, 24.3 per cent less in 1929 than in 1925. The proportionate share of the larger cities in the total increased during the period under consideration, while that of the smaller cities decreased. Class A cities laid 29.2 per cent of the total in 1925 and 37.9 per cent in 1929. Class B cities laid 8.2 per cent of the whole in 1925 and 10.1 per cent in 1929. The share of Class C cities in the total decreased from 17.3 per cent in 1925 to 14.6 per cent in 1929; that of Class D cities, from 17.0 per cent in 1925 to 15.0 per cent in 1929; and that of Class E cities from 28.3 per cent in 1925 to 22.4 per cent in 1929.

Sheet Asphalt. - Seven-eighths of all sheet asphalt laid in the 201 cities was laid east of the Mississippi River and Lake Michigan. Four-fifths (79.6 per cent) of the total was laid in the Northeastern district and 8.3 per cent, in the Southeastern district. West of the Mississippi River only 7.6 per cent of the national total was laid in the Pacific-Rocky Mountain district, only 2.6 per cent in the North-Central district, and only 2.0 per cent in the Southwestern district.

Sheet asphalt street pavement laid in 201 cities, 1925-1929, by geographic districts
(In square yards)

District	1925	1926	1927	1928	1929
Northeastern	14,634,643	15,740,390	16,798,562	15,813,943	14,047,248
Southeastern	1,720,065	1,508,629	2,431,319	1,134,891	1,410,150
Southwestern	491,412	479,351	304,213	459,133	182,575
North Central	639,058	585,026	374,612	439,726	437,831
Pacific-Rocky Mountain	1,290,8[illegible]	1,375,977	1,490,036	1,287,432	1,854,291
Total	18,776,011	19,489,373	21,404,872	19,135,125	17,932,095

Most of the sheet asphalt was laid in the larger cities. Nearly half (49.7 per cent) of the total area of streets surfaced with sheet asphalt from 1925 to 1929 was in the five Class A cities with populations of more than 1,000,000. Eight Class B cities, containing from 500,000 to 1,000,000 residents, put down 10.8 per cent of the total. An almost equal proportion, 11.1 per cent, was in the 24 Class C cities, with populations of 250,000 to 500,000; and a slightly larger fraction, 12.0 per cent, in 51 Class D cities, whose populations ranged between 100,000 and 200,000. The remaining 16.5 per cent was distributed among 113 municipalities of Class E, with less than 100,000 inhabitants each.

Sheet asphalt street pavement laid in 201 cities, 1925-1929, by population groups
(In square yards)

	1925	1926	1927	1928	1929
Class A ...	8,636,971	9,724,850	10,436,770	9,593,608	9,634,009
Class B ...	1,734,716	2,038,197	2,294,740	2,192,673	2,224,408
Class C ...	2,492,968	1,965,890	2,131,229	2,243,750	1,892,238
Class D ...	2,271,972	2,511,297	2,859,013	2,270,972	1,672,470
Class E ...	3,639,384	3,249,139	3,703,120	2,829,122	2,508,970
Total ..	18,776,011	19,489,373	21,404,872	19,135,125	17,932,095

Although the area of sheet asphalt laid was 14.0 per cent greater in 1927 than in 1925, the yardage laid in 1929 was 16.2 per cent less than in 1927, the peak year. The largest relative decreases took place in the smaller cities. In Class D cities 26.4 per cent less sheet asphalt was laid in 1929 than in 1925, and in Class E cities 31.1 per cent less. In Class B cities, on the other hand, the yardage of sheet asphalt laid in 1929 exceeded that laid in 1925 by 28.2 per cent. In Class A cities the yardage of sheet asphalt pavement followed the

general national trend of all bituminous paving and of all street pavements, rising to a peak in 1927 and decreasing in 1928 and 1929. Yet the yardage of sheet asphalt laid in Class A cities in 1929 was 11.6 per cent larger than that laid in 1925. As a result of these varying rates of decrease, the ratio of sheet asphalt laid in Class A cities to the total increased from 46.0 per cent in 1925 to 53.7 per cent in 1929, and in Class B cities from 9.2 per cent in 1925 to 12.4 per cent in 1929. On the other hand, the share of Class C cities decreased from 13.3 per cent of the whole in 1925 to 10.6 per cent in 1929, of Class D cities from 12.1 per cent in 1925 to 9.3 per cent in 1929, and of Class E cities from 19.4 per cent in 1925 to 14.0 per cent in 1929.

From the geographic point of view, the largest decreases in the use of sheet asphalt for street paving took place in interior United States west of the Mississippi River and Lake Michigan and east of the Rocky Mountains. In the Southwestern district 62.8 per cent less sheet asphalt was laid in 1929 than in 1925; and in the North-Central district, 31.5 per cent less. In the Southeastern district sheet asphalt paving decreased 18.0 per cent during the same period.

The use of sheet asphalt for street paving tended to be concentrated to a greater extent in the northeastern quarter of the country and on the Pacific coast. Cities of the Northeastern district laid 77.8 per cent of the total sheet asphalt in 1925 and 78.1 per cent in 1929; and cities of the Pacific-Rocky Mountain district, 6.9 per cent of the total in 1925 and 10.3 per cent in 1929.

Asphaltic Concrete. - Only two-fifths of the asphaltic concrete laid from 1925 to 1929, inclusive, in the 201 cities comprised in this survey was put down east of the Mississippi River. Less than one-third (30.7 per cent) was laid in the Northeastern district, although this district laid 61.7 per cent of the total bituminous pavement. In the Southeastern district 8.8 per cent of the total asphaltic concrete was put down, which corresponded fairly well with the 8.9 per cent of all bituminous pavements laid. West of the Mississippi River more than one-third (36.0 per cent) was put down in the Pacific-Rocky Mountain district, although this district laid only 16.3 per cent of all bituminous paving. In the Southwestern district, the proportion of the total asphaltic concrete laid, 18.1 per cent, was almost double the proportion, 9.2 per cent, of all asphaltic pavements. Only 6.4 per cent of the total asphaltic concrete was laid in the North Central district, which laid only 3.9 per cent of the total bituminous pavement.

Asphaltic concrete street pavement laid in 201 cities, 1925-1929, by geographic districts
(In square yards)

District	1925	1926	1927	1928	1929
Northeastern	3,230,736	3,263,644	3,379,085	2,951,764	2,460,161
Southeastern	783,777	1,019,797	886,503	921,782	760,610
Southwestern	1,661,126	1,932,728	2,067,804	1,738,035	1,588,957
North-Central	1,030,836	695,946	720,729	401,633	352,941
Pacific-Rocky Mountain	3,138,667	5,276,585	2,705,159	3,716,472	3,065,908
Total	9,845,142	12,188,700	9,759,280	9,729,686	8,228,577

The use of asphaltic concrete for city street paving increased 23.8 per cent, from 9,845,142 square yards in 1925 to a maximum of 12,188,700 square yards in 1926, then decreased until only 8,228,577 square yards was laid in 1929, or 16.4 per cent less than in 1925. Except in the North Central district, where the laying of asphaltic concrete dropped 65.8 per cent from 1925 to 1929, this trend was general throughout the nation. In the Pacific-Rocky Mountain district the changes were more exaggerated.

More than three-fourths of the asphaltic concrete was laid from 1925 to 1929 in cities of less than 500,000 population, and more than five-ninths of it in cities of less than 250,000 population. The five Class A cities with populations of more than 1,000,000, laid only 13.5 per cent of the total; and the eight Class B cities, ranging in population from 500,000 to 1,000,000, only 9.0 per cent. The 24 Class C cities laid 21.6 per cent of the asphaltic concrete, although they laid only 14.7 per cent of the total bituminous pavement. Similarly, the 51 Class D cities, containing 100,000 to 250,000 inhabitants, laid 23.6 per cent of the asphaltic concrete, but only 17.1 per cent of the total bituminous pavement. Nearly one-third (32.4 per cent) of the asphaltic concrete was laid in the 113 Class D cities of less than 100,000 population, although these laid only 26.8 per cent of the total bituminous pavement.

Asphaltic concrete street pavement laid in 201 cities, 1925-1929, by population groups
(In square yards)

	1925	1926	1927	1928	1929
Class A ...	840,157	1,430,852	1,159,674	1,771,392	1,496,198
Class B ...	859,463	956,510	886,133	884,842	885,421
Class C ...	2,591,872	2,717,049	2,052,788	1,545,745	1,837,570
Class D ...	2,614,112	2,403,611	2,375,969	2,476,083	1,856,345
Class E ...	2,939,538	4,680,678	3,284,716	3,051,624	2,153,043
Total ..	9,845,142	12,188,700	9,759,280	9,729,686	8,228,577

Asphaltic Macadam. - The use of asphaltic penetration macadam to surface city streets is localized largely in the northeastern quarter of the United States. Six-sevenths (85.5 per cent) of the total area of macadam treated with asphalt by the penetration method put down from 1925 to 1929, inclusive, in the 201 cities was laid north of the Potomac and Ohio Rivers and east of the Mississippi River and Lake Michigan, chiefly in Massachusetts, Connecticut, Rhode Island, and Illinois. Of the remaining one-seventh the greater part was spread in the North-Central district lying west of Lake Michigan and the Mississippi River, chiefly in Wisconsin.

Asphaltic macadam street pavement laid in 201 cities, 1925-1929, by geographic districts
(In square yards)

District	1925	1926	1927	1928	1929
Northeastern	1,157,272	1,261,976	1,113,494	1,464,102	2,292,282
Southeastern	41,217	31,680	16,250	69,628	41,193
Southwestern	--	--	--	--	151,192
North-Central	148,446	209,052	240,092	185,589	101,566
Pacific-Rocky Mountain	--	--	--	--	587
Total	1,346,935	1,502,708	1,369,836	1,719,319	2,586,820

Even more than asphaltic concrete, the use of asphaltic penetration macadam is characteristic of the smaller cities and towns. Nearly four-fifths of the asphaltic macadam laid in the 201 cities canvassed was put down in municipalities having populations of less than 250,000. Nearly one-half (49.4 per cent) was laid in Class E cities with less than 100,000 residents; and three-tenths (29.6 per cent) in Class D cities numbering from 100,000 to 250,000 inhabitants. Only one-eighth (12.2 per cent) of the total was laid in the five Class A cities with over 1,000,000 population; and only 8.7 per cent in the eight Class B cities ranging in population from 500,000 to 1,000,000.

Asphaltic macadam street pavement laid in 201 cities, 1925-1929, by population groups
(In square yards)

	1925	1926	1927	1928	1929
Class A ...	126,851	190,827	85,462	133,597	503,789
Class B ...	122,385	186,501	149,814	193,495	88,362
Class C ...	2,003	---	---	5,931	---
Class D ...	356,142	475,702	278,859	549,633	868,316
Class E ...	739,554	649,678	855,701	836,663	1,126,353
Total ..	1,346,935	1,502,708	1,369,836	1,719,319	2,586,820

Asphalt Block

All of the asphalt block was laid east of the Mississippi River. More than seven-tenths (71.9 per cent) of the total was laid in the Southeastern district, lying south of the Potomac and Ohio Rivers, all in Florida, and only 28.1 per cent in the Northeastern district, lying north of these rivers, almost entirely in New York and Ohio.

Nearly three-fifths of all asphalt block set in the 201 cities was laid in Class E municipalities of less than 100,000 inhabitants, and 27 per cent in Class D cities, with populations of 100,000 to 250,000. The remaining one-seventh was laid in the city of New York.

Native Rock Asphalt. - Nearly five-sixths (82.5 per cent) of the total area surfaced with native rock asphalt in the 201 cities during the five years ended December 31, 1929, was laid in the Southwestern district, chiefly in Texas, Louisiana, Oklahoma, and Kansas. Only 9.2 per cent was laid in the Southeastern district, chiefly in Alabama, Tennessee, and West Virginia, and only 8.3 per cent in the Northeastern district, chiefly in Illinois and Ohio.

Native rock asphalt street pavement laid in 201 cities, 1925-1929, by geographic districts
(In square yards)

	1925	1926	1927	1928	1929
Northeastern	88,209	129,109	78,756	111,617	9,564
Southeastern	67,922	39,926	75,885	103,344	172,096
Southwestern	696,594	645,409	1,008,694	884,979	905,120
North-Central	---	---	---	---	---
Pacific-Rocky Mountain	---	---	---	---	---
Total...........	852,725	814,444	1,163,335	1,099,940	1,086,780

Seven-tenths of the rock asphalt was laid in cities of middle size during the period from 1925 to the end of 1929. Over five-ninths (57.1 per cent) was laid in Class C cities, with populations ranging between 250,000 and 500,000; and 13.5 per cent in Class D cities, containing from 100,000 to 250,000 inhabitants. Class E cities, which numbered less than 100,000 residents, laid 27.7 per cent.

Native rock asphalt street pavement laid in 201 cities, 1925-1929, by population groups
(In square yards)

	1925	1926	1927	1928	1929
Class A	---	---	31,724	49,918	---
Class B	4,570	3,690	---	---	---
Class C	591,754	427,884	540,141	505,789	796,924
Class D	79,096	10,484	118,210	197,794	273,943
Class E	177,305	372,386	473,260	346,439	15,913
Total	852,725	814,444	1,163,335	1,099,940	1,086,780

Miscellaneous Bituminous Pavements. - Three-fifths of the total area of miscellaneous bituminous types of surfacing laid in the 201 cities was put down in the Northeastern district. Over three-tenths (31.1 per cent) was laid in the Pacific-Rocky Mountain district, especially in California. Only 5.8 per cent of the total was laid in the Southwestern district, and only 3.2 per cent in the Southeastern district.

Miscellaneous bituminous street pavements laid in 201 cities, 1925-1929, by geographic districts
(In square yards)

District	1925	1926	1927	1928	1929
Northeastern	877,808	958,502	907,272	898,432	1,110,884
Southeastern	104,888	121,408	10,188	--	20,496
Southwestern	52,529	41,457	117,026	91,803	153,534
North-Central	--	--	--	--	1,783
Pacific-Rocky Mountain	765,357	398,445	581,203	305,235	413,179
Total	1,800,582	1,519,812	1,615,689	1,295,470	1,699,876

Five-sixths (82.9 per cent) of the area of miscellaneous bituminous types of pavement laid in the 201 cities from January 1, 1925, to December 31, 1929, was put down in Class E cities of less than 100,000 inhabitants each. Six per cent was laid in Class D cities of 100,000 to 250,000 residents, 6.6 per cent in Class C cities of 250,000 to 500,000 population, and 4.4 per cent in Class A cities numbering over 1,000,000 inhabitants.

The national trend in the use of Portland-cement concrete for street paving was paralleled fairly closely by the Pacific-Rocky Mountain district and the Northeastern district, and in a general way by the Southeastern district and the North-Central district. In the Southwestern district the use of Portland cement concrete for street paving increased moderately and fairly evenly up to 1928, but rose sharply in 1929.

From 19.1 per cent in 1925 the share of Portland-cement concrete in the total area of paving laid grew steadily to 29.7 per cent in 1928, but dropped to 28.2 per cent in 1929. The highest proportionate gain over the whole period took place in the Northeastern district, where the share of Portland-cement concrete in the total rose from 9.9 per cent in 1925 to 17.1 per cent in 1929. In the Southeastern district the ratio of Portland-cement concrete to the whole paving increased from 17.9 per cent in 1925 to 26.4 per cent in 1929, after a brief drop in 1926. In the Southwestern district the proportion of Portland-cement concrete to the total pavement, after varying from 1925 to 1928 between 21.5 and 23.5 per cent, rose in 1929 to 28.0 per cent. In the North-Central district a slow but fairly steady increase brought the percentage of Portland-cement concrete from 49.6 per cent of the total pavement in 1925 to 58.8 per cent in 1929. West of the Rocky Mountains the share of Portland-cement concrete in the total paving increased from 34.6 per cent in 1925 to 58.6 per cent in 1928, but dropped to 49.6 per cent in 1929.

In general, the use of Portland-cement concrete for street paving is distributed among cities of various sizes roughly in proportion to the total pavement laid. For instance, Class A cities laid 30.0 per cent of the Portland-cement concrete and 31.5 per cent of the total paving laid in the 201 municipalities. Class B cities laid 7.8 per cent of the Portland-cement concrete and 9.4 per cent of the total pavement. More variation is shown in the medium-sized and smaller cities. In Class C cities 22.0 per cent of the Portland-cement concrete was laid as compared with 16.4 per cent of the total pavement. Class D cities laid 18.8 per cent of the Portland-cement concrete and 16.7 per cent of the total pavement; and Class E cities, 21.4 per cent of the Portland-cement concrete and 25.9 per cent of the total pavement.

Portland-cement concrete street pavement laid in 201 cities, 1925-1929, by population groups
(In square yards)

	1925	1926	1927	1928	1929
Class A	1,873,737	2,776,423	5,060,395	5,650,751	4,421,605
Class B	604,374	963,141	1,223,780	1,243,799	1,132,310
Class C	2,249,853	2,518,514	3,426,845	3,385,838	2,959,896
Class D	2,074,780	2,371,868	2,594,545	2,679,039	2,674,530
Class E	2,247,310	2,777,905	3,603,392	2,929,717	2,588,740
Total ...	9,050,054	11,407,851	15,908,957	15,889,144	13,777,081

The trend in the use of Portland-cement concrete was fairly generally distributed throughout cities of all sizes. In Class A and Class B cities the rate of increase up to 1928 and the rate of decrease in 1929 were sharper than

Miscellaneous bituminous street pavements laid in 201 cities, 1925-1929, by population groups
(In square yards)

	1925	1926	1927	1928	1929
Class A	--	49,903	111,581	144,376	327,277
Class B	--	--	--	--	--
Class C	66,588	59,353	160,876	141,034	92,000
Class D	134,133	278,373	42,177	8,009	18,385
Class E	1,599,861	1,132,183	1,301,055	1,002,051	1,262,214
Total	1,800,582	1,519,812	1,615,689	1,295,470	1,699,876

Portland-Cement Concrete

The total street area surfaced with Portland-cement concrete increased 26.1 per cent, from 9,050,054 square yards in 1925 to 11,407,851 square yards in 1926, 39.5 per cent to 15,908,957 square yards in 1927, but declined 0.1 per cent to 15,889,144 square yards in 1928, and dropped 13.3 per cent to 13,777,081 square yards in 1929. From 1925 to 1927 the yardage of Portland-cement concrete increased 75.8 per cent, while the total area of street pavement laid in the 201 cities increased only 22.7 per cent. The decrease of 0.1 per cent in Portland-cement pavement from 1927 to 1928 coincided with a decrease of 7.5 per cent in the yardage of all types of pavement, but the decrease of 13.3 per cent in the area of Portland-cement concrete laid from 1928 to 1929 was greater than the decline of 8.7 per cent in the total pavement laid.

Three-fifths of the Portland-cement concrete pavement laid in the 201 cities from 1925 to 1929, inclusive, was put down west of the Mississippi River and Lake Michigan, although only 35.8 per cent of the total pavement was laid in this vast region. Nearly two-fifths (39.3 per cent) was laid in 22 cities of the Pacific-Rocky Mountain district, although these cities contained only 10.6 per cent of the total population and laid only 20.7 per cent of the total pavement. More than one-fourth (27.4 per cent) of the total was laid in California, although this State laid only one-sixth (16.1 per cent) of the total street pavement. Cities of the North-Central district laid 13.6 per cent of the Portland-cement concrete, but only 6.4 per cent of the total pavement; and cities of the Southwestern district, 8.1 per cent of the Portland-cement concrete, in comparison with 8.7 per cent of the total pavement. East of the Mississippi River, nearly one-third (32.0 per cent) was laid in the Northeastern district, which put down 55.0 per cent of the total pavement; and 7.0 per cent in the Southeastern district, which laid 9.2 per cent of the total pavement.

Portland-cement concrete street pavement laid in 201 cities, 1925-1929, by geographic districts
(In square yards)

District	1925	1926	1927	1928	1929
Northeastern	2,636,091	3,558,965	5,458,370	4,838,113	4,655,813
Southeastern	846,621	809,962	1,023,251	992,050	955,223
Southwestern	908,577	1,040,725	1,064,695	1,030,400	1,289,727
North-Central	1,910,141	1,847,964	2,056,513	1,571,858	1,621,428
Pacific-Rocky Mountain	2,748,624	4,150,235	6,306,128	7,456,723	5,254,890
Total	9,050,054	11,407,851	15,908,957	15,889,144	13,777,081

the general trend and in Class C cities they were remarkably parallel to it. In Class D cities there was a fairly even rate of increase from 1925 to 1929. In Class E cities, however, the peak was reached in 1927 and the decline from 1927 to 1929 was sharper than the general decline.

Brick, Block, and Stone

The area of city streets surfaced with vitrified brick, wood block, stone block or slab, increased 12.2 per cent, from 5,084,180 square yards in 1925 to 5,703,273 square yards in 1927, but dropped 22.4 per cent to 4,412,155 square yards in 1928 and 21.9 per cent to 3,413,019 square yards in 1929. The rate of increase for brick, block, and stone pavements from 1925 to 1927 was more than one-third less than the general rate of increase in paving, while the rates of decrease from 1927 to 1929 were more than three times the general rate of decrease in all paving. The share of brick, block, and stone in paving the city streets decreased steadily from 10.8 per cent in 1925 to 7.0 per cent in 1929.

Nine-tenths of all brick, block, and stone street pavements laid in the 201 cities was put down east of the Mississippi River and Lake Michigan. Nearly three-fourths (74.1 per cent) was laid north of the Potomac and Ohio Rivers, and 15.9 per cent in the Southeastern States. West of the Mississippi River and Lake Michigan, only 6.4 per cent of the national total was laid in the Southwestern district, only 3.6 per cent in the North-Central district, and only 0.1 per cent in the Pacific-Rocky Mountain district.

Brick, block, and stone street pavements laid in 201 cities, 1925-1929, by geographic districts
(In square yards)

District	1925	1926	1927	1928	1929
Northeastern	3,844,391	3,607,570	4,386,283	3,487,052	2,624,952
Southeastern	873,768	1,584,816	750,965	425,203	205,749
Southwestern	225,983	288,030	380,280	305,227	338,633
North-Central	125,377	133,041	182,722	193,720	240,081
Pacific-Rocky Mountain	14,661	1,865	3,023	953	3,604
Total	5,084,180	5,615,322	5,703,273	4,412,155	3,413,019

The sharpest decreases in the use of brick, block, and stone pavements occurred in the Southeastern district, where only 23.6 per cent of the area surfaced with brick, block, and stone in 1925 was so paved in 1929; and in the Northeastern district, where only 68.3 per cent of the area paved with brick, block, and stone in 1925 was so paved in 1929. In the Southwestern district, however, the area paved with brick, block, and stone was 49.9 per cent larger in 1929 than in 1925; and in the North-Central district, 91.5 per cent larger.

Brick, block, and stone pavements were laid chiefly in the largest and in the smallest of the 201 cities. Three-tenths (29.9 per cent) of the total yardage of brick, block, and stone street pavement was laid in the five Class A

cities, chiefly in New York, and 30.2 per cent in the 113 Class E cities. Eight Class B cities laid 13.9 per cent of the total; 24 Class C cities, 13.5 per cent; and 51 Class D cities, 12.5 per cent.

Brick, block, and stone street pavement laid in 201 cities, 1925-1929, by population groups
(In square yards)

	1925	1926	1927	1928	1929
Class A	1,539,116	1,651,378	1,855,122	1,303,068	885,675
Class B	758,067	569,902	770,553	596,968	681,389
Class C	527,737	574,532	792,642	656,745	717,946
Class D	548,639	1,017,327	487,495	488,518	492,261
Class E	1,710,621	1,802,183	1,797,461	1,366,856	635,748
Total	5,084,180	5,615,322	5,703,273	4,412,155	3,413,019

From the standpoint of size, the most marked decreases in the use of brick, block, and stone pavements took place in the largest and in the smallest cities. In Class A cities the use of brick, block, and stone to pave city streets decreased 42.4 per cent from 1925 to 1929 and in Class E cities 62.8 per cent. In Class B cities the street area surfaced with brick, block, and stone was only 10.1 per cent less than 1925, and in Class D cities only 10.3 per cent less. On the other hand, in Class C cities the area surfaced with brick, block, and stone was 36.0 per cent larger.

DISCUSSION BY DISTRICTS

Northeastern District

Five-ninths (55.2 per cent) of all hard-surfaced pavements laid in the 201 cities comprised in this survey from 1925 to 1929, inclusive, were laid in the Northeastern district lying north of the Potomac and Ohio Rivers and east of the Mississippi River and Lake Michigan. The 113 cities of this district from which data were received contained 68.1 per cent of the total urban population covered by this survey. Nearly 49 per cent of its inhabitants live in cities of over 40,000 residents. Four of the five Class A cities of over 1,000,000 population, five of the eight Class B cities of 500,000 to 1,000,000 population, and 10 of the 24 Class C cities of 250,000 to 500,000 population lie in this district. Although the Northeastern district comprises only about 12.4 per cent of the area of continental United States, it contained in 1930 over 48 per cent of the total population and 64.4 per cent of the urban population living in communities of over 40,000 inhabitants. In it is conducted seven-tenths of the manufacturing of the Nation, as measured by the number of wage-earners, the value of products, and the primary horsepower employed.

Street pavement laid in Northeastern district, 1925-1929, by types
(In square yards)

	1925	1926	1927	1928	1929
Sheet asphalt	14,634,643	15,740,390	16,798,662	15,813,943	14,047,248
Asphaltic concrete	3,230,736	3,263,644	3,379,085	2,951,764	2,460,161
Asphaltic macadam	1,157,272	1,261,976	1,113,494	1,464,102	2,292,282
Asphalt block	127,674	180,831	229,381	136,484	77,885
Natural rock asphalt	88,209	129,109	78,756	111,617	9,564
Other asphaltic types of paving	877,808	958,502	907,272	898,432	1,110,884
Tar macadam	114,433	39,055	27,917	10,099	10,000
Total bituminous	20,230,775	21,573,507	22,534,567	21,386,441	20,008,024
Portland-cement concrete	2,636,091	3,558,965	5,458,370	4,838,113	4,655,813
Brick, block, and stone	3,844,391	3,607,570	4,386,283	3,487,052	2,624,952
Total	26,711,257	28,740,042	32,379,220	29,711,606	27,288,789

Nearly three-fourths (73.0 per cent) of the street area paved in the Northeastern district from 1925 to 1929, inclusive, was surfaced with bituminous mixtures. Sheet asphalt constituted 72.9 per cent of all the bituminous pavements and 53.2 per cent of all hard-surfaced pavements laid in the district during that period. Five States, New York, Pennsylvania, Ohio, Michigan, and Illinois, laid nearly five-sixths of the sheet asphalt. Asphaltic concrete formed only 14.5 per cent of the total bituminous pavement; asphaltic penetration macadam, 6.9 per cent, miscellaneous bituminous types, 4.5 per cent of the bituminous surface. Only small percentages of asphalt block, of natural rock asphalt, and of coal-tar macadam were laid.

Although the total area surfaced with bituminous mixtures in the Northeastern district rose and fell with the increases and decreases in all paving from 1925 to 1929, it declined 1.1 per cent in yardage over the five-year period and dropped in relative importance from 75.7 per cent in 1925 to 73.3 per cent in 1929. A similar trend occurred in the laying of sheet asphalt. The decline in both the yardage and the relative importance of asphaltic concrete over the five-year period was largely compensated by an increase in the laying of asphaltic penetration macadam, especially in 1928 and 1929. Miscellaneous bituminous pavements generally rose in 1926, declined during 1927 and 1928 both in yardage and in relative importance, but rose in 1929 to a yardage 26.6 per cent larger than in 1925.

About 14.5 per cent of the total street pavement laid in the Northeast district from 1925 to 1929 consisted of Portland-cement concrete. During this period, the street area paved with Portland-cement concrete more than doubled, increasing from 2,636,091 square yards in 1925 to 5,458,370 square yards in 1927, but dropped to 4,838,113 square yards in 1928 and to 4,655,813 square yards in 1929. The rate of increase from 1925 to 1927, 107.1 per cent, was more than five times the general increase in paving in the Northeastern district during that period. On the other hand, the decrease of 14.7 per cent from 1927 to 1929 fell short of the rate of general decrease in paving this district, 15.9 per cent. Consequently, the proportion of Portland-cement concrete to total pavement increased fairly steadily from 9.9 per cent in 1925 to 17.1 per cent in 1929.

About one-eighth (12.4 per cent) of the total area of street pavement laid in the Northeastern district consisted of vitrified brick, stone block or slab, or wood block.

The area of street pavement surfaced with brick, block, and stone in the Northeastern district increased from 3,844,391 square yards in 1925 to 4,386,283 square yards in 1927, but decreased to 3,487,052 square yards in 1928 and to 2,624,952 square yards in 1929. The increase of 14.3 per cent in brick, block, and stone pavement laid from 1925 to 1927 fell one-quarter short of the general increase of 21.2 per cent in all pavement laid in the Northeastern district, while the decrease of 20.5 per cent from 1927 to 1928 was two and one-half times the general decrease of 8.3 per cent in all paving laid in the Northeastern district; and the decrease of 24.7 per cent from 1928 to 1929 was nearly three times the general decrease of 8.5 per cent in all paving. Consequently, the share of brick, block, and stone pavements in the total street pavement laid in the Northeastern district decreased from 14.4 per cent in 1925, 12.6 per cent in 1926, and 13.5 per cent in 1927 to 11.7 per cent in 1928 and 9.6 per cent in 1929.

Southeastern District

Only 9.2 per cent of the total pavement comprised in this survey was laid in the Southeastern district, lying south of the Potomac and Ohio Rivers and east of the Mississippi and Pearl Rivers. Although the Southeastern district comprises 14.2 per cent of the total area of continental United States and contained 19.0 per cent of its total population in 1930, only 7.5 per cent of the total urban population contained in the 201 cities lived in the 28 cities which represent the Southeastern States. Sixteen of these cities have less than 100,000 inhabitants, nine of them from 100,000 to 250,000, and three of them from 250,000 to 500,000 each. None of the Class A cities, with populations exceeding 1,000,000 or Class B cities of 500,000 to 1,000,000 inhabitants, are situated in this area. Whereas 36.7 per cent of the population of continental United States in 1930 lived in cities and towns of over 40,000 inhabitants, only 15.6 per cent of the inhabitants of the Southeastern district in 1930 lived in communities of over 40,000 persons. The economic basis of the Southeastern States lies in agriculture rather than in manufacture, although manufacturing, especially of textiles, tobacco, steel, and wood products has increased rapidly in recent years.

Street paving laid in the Southeastern district increased 20.2 per cent, from 4,727,107 square yards in 1925 to 5,696,882 square yards in 1926, and 4.6 per cent to 5,956,072 square yards in 1927, then decreased 37.4 per cent to 3,714,401 square yards in 1928 and 2.5 per cent to 3,621,420 square yards in 1929. The most marked decrease took place in Florida where the yardage of all street pavement laid fell from a peak of 2,815,074 square yards in 1926 to 2,185,819 square yards in 1927, to 891,783 square yards in 1928, and to 278,859 square yards in 1929. Decreases during 1928 and 1929 in the street pavement laid in Georgia, North Carolina, South Carolina, and Kentucky more than offset the increases in Alabama, Tennessee, and Virginia.

Street-pavement laid in the Southeastern district, 1925-1929, by types of paving
(In square yards).

Type	1925	1926	1927	1928	1929
Sheet asphalt............	1,720,065	1,308,629	2,431,319	1,134,891	1,410,150
Asphaltic concrete	783,777	1,019,797	886,503	921,782	760,610
Asphaltic macadam	41,217	31,680	16,250	69,628	41,193
Asphalt block	288,849	780,664	761,711	67,503	55,903
Natural rock asphalt....	67,922	39,926	75,885	103,344	172,096
Other asphaltic types of paving	104,888	121,408	10,188	- -	20,496
Total bituminous ...	3,006,718	3,302,104	4,181,856	2,297,148	2,460,448
Portland-cement concrete	846,621	809,962	1,023,251	992,050	955,223
Brick, block, and stone	873,768	1,584,816	750,965	425,203	205,749
Total..............	4,727,107	5,696,882	5,956,072	3,714,401	3,621,420

The total yardage paved with bituminous mixtures in the Southeastern district increased from 3,006,718 square yards in 1925 to 4,181,856 square yards in 1927, but dropped to 2,297,148 square yards in 1928, and rose again to 2,460,448 square yards in 1929. The increase of 39.1 per cent in the area paved with bituminous mixtures from 1925 to 1927 exceeded by one-half the increase of 26.0 per cent in all pavement laid during that period. On the other hand, the decrease of 45.1 per cent in bituminous pavement laid from 1927 to 1928 was greater than the decrease of 37.6 per cent in the total pavement laid. In 1929 the bituminous pavement laid was 7.1 per cent greater than in 1928, although the total pavement laid in the Southeastern district was 2.5 per cent less. Accordingly, the ratio of the bituminous pavements laid to the total pavement laid increased from 63.6 per cent in 1925 to 70.2 per cent in 1927, dropped to 61.8 per cent in 1928 and rose to 67.9 per cent in 1929.

Although the yardage of Portland-cement concrete laid in the Southeastern States fluctuated from year to year, the net result was an increase of 12.8 per cent in 1929 over the yardage laid in 1925, while the total pavement laid in the Southeastern district decreased 23.4 per cent over the same period. The ratio of Portland-cement concrete to the total pavement varied as follows: 17.9 per cent in 1925, 14.2 per cent in 1926, 17.2 per cent in 1927, 26.7 per cent in 1928, and 26.4 per cent in 1929.

The area paved with brick, block, and stone in the Southeastern States increased from 873,768 square yards in 1925 to 1,584,816 square yards in 1926, then dropped at an increasing rate of decline to 205,749 square yards in 1929. More than five-eights of the total was laid in Florida, and consisted chiefly of vitrified brick.

More than half (52.5 per cent) of all bituminous pavements laid from 1925 to 1929, inclusive, in the 28 cities which represent the Southeastern district consisted of sheet asphalt. Asphaltic concrete of various types constituted 28.7 per cent of the whole, and asphalt block 12.8 per cent. Natural rock asphalt was used to surface only 3.0 per cent of the total area paved with bituminous compounds; asphaltic penetration macadam, only 1.3 per cent; and miscellaneous bituminous types, 1.7 per cent.

The yardage of sheet asphalt pavements laid in the Southeastern district decreased 23.5 per cent, from 1,720,065 square yards in 1925 to 1,308,629 square yards in 1926, increased 85.8 per cent to 2,431,319 square yards in 1927, decreased 53.3 per cent to 1,134,891 square yards in 1928, and rose 24.2 per cent to 1,410,150 square yards in 1929. Sheet asphalt formed 57.2 per cent of the total bituminous pavements in 1925; 39.6 per cent in 1926; 58.1 per cent in 1927; 49.4 per cent in 1928; and 57.3 per cent in 1929.

The laying of asphaltic concrete in the Southeastern district was subject to less fluctuation than that of sheet asphalt. From 783,777 square yards in 1925 it rose to 1,019,797 square yards in 1926; fell to 886,503 square yards in 1927, rose again to 921,782 square yards in 1928, and dropped to 760,610 square yards in 1929. Three-fifths of the yardage of asphaltic concrete was laid in Alabama; the rest was variously apportioned among the remaining nine Southeastern States.

The area surfaced with asphalt block in the Southeastern district increased from 288,849 square yards in 1925 to 780,664 square yards in 1926 and 761,711 square yards in 1927, but dropped to 67,503 square yards in 1928 and 55,903 square yards in 1929. All the asphalt block was laid in Florida.

Southwestern District

Only 8.7 per cent of all hard-surfaced pavements put down in the 201 cities was laid from 1925 to 1929 in the Southwestern district, which lies west of Mississippi and Pearl Rivers and south of St. Louis, Kansas City, Wichita, Amarillo, and El Paso. The 20 cities of this district from which data were received contained 8.8 per cent of the total urban population covered by this survey. Nine of the 20 cities have less than 100,000 inhabitants, five contain from 100,000 to 250,000, five include from 250,000 to 500,000, and only one has more than 500,000 residents. None of the Class A cities of over 1,000,000 population is included in this district. Although the Southwestern district comprises 16.8 per cent of the area of continental United States, it contained in 1930 only 14.4 per cent of the total population and only 8.9 per cent of the urban population dwelling in communities of over 40,000 persons. Even more than the Southeastern district, its economy is based on the production of raw materials, agricultural, and mineral.

Street pavement laid in Southwestern district, 1925-1929, by types of paving
(In square yards)

Type	1925	1926	1927	1928	1929
Sheet asphalt	491,412	479,351	304,213	459,133	182,575
Asphaltic concrete	1,661,126	1,932,728	2,067,804	1,738,035	1,588,957
Asphaltic macadam	--	--	--	--	151,192
Asphalt block	--	--	--	--	--
Natural rock asphalt	696,594	645,409	1,008,694	884,979	905,120
Other asphaltic types of paving	52,529	41,457	117,026	91,803	153,534
Total bituminous	2,901,661	3,098,945	3,497,737	3,173,950	2,981,378
Portland-cement concrete	908,577	1,040,725	1,064,695	1,030,400	1,289,727
Brick, block, and stone..	225,983	288,030	380,280	305,227	338,633
Total	4,036,221	4,427,700	4,942,712	4,509,577	4,609,738

The total area of street pavement laid in the Southwestern district increased from 4,036,221 square yards in 1925 to a maximum of 4,942,712 square yards in 1927, dropped to 4,509,577 square yards in 1928, and rose to 4,609,738 square yards in 1929. Though the yardage laid in 1929 was 6.7 per cent less than that laid during the peak year of 1927, it was 14.2 per cent larger than that laid in 1925.

From 1925 to 1928 the area of bituminous pavements laid in the Southwestern district kept pace with the general increases and decreases in all types of paving. The area of bituminous paving rose 20.5 per cent from 2,901,661 square yards in 1925 to a peak of 3,497,737 square yards in 1927, but dropped 9.3 per cent to 3,173,950 square yards in 1928. In 1929, however, the total yardage of all bituminous pavements, 2,981,378 square yards, was 6.1 per cent less than in 1928, although the total area of all street pavement laid in 1920 was 2.2 per cent greater than in the preceding year. The ratio of bituminous pavements to the total pavement, which had fluctuated between 70 and 72 per cent from 1925 to 1928, dropped in 1929 to 64.7 per cent.

More than five-ninths (57.4 per cent) of all bituminous paving laid in the Southwestern district consisted of asphaltic concrete, and more than one-fourth (26.5 per cent) of natural rock asphalt. Five-sixths of the natural rock asphalt was laid in Texas and more than one-tenth in Louisiana. Sheet asphalt constituted only 12.2 per cent of the total bituminous paving; and miscellaneous bituminous types, 2.9 per cent.

The yardage of Portland-cement concrete pavement in the Southwestern district increased from 908,577 square yards in 1925 to 1,040,725 square yards in 1926 and 1,064,695 square yards in 1927, dropped to 1,030,400 square yards in 1928, and rose to 1,289,727 square yards in 1929. From 1925 to 1928 the ratio of Portland-cement concrete to total pavement varied between 21.5 and 23.5 per cent. In 1929, however, Portland-cement concrete formed 28.0 per cent of the whole area of street pavement.

The street area surfaced with brick, block, and stone in the Southwestern district increased both in yardage and in relative importance. From 225,983 square yards in 1925 it increased to 380,280 square yards in 1927, dropped to 305,227 square yards in 1928, and rose to 338,633 square yards in 1929. From 5.6 per cent of the total pavement in 1925, its ratio rose to 6.5 per cent in 1926, to 7.7 per cent in 1927, fell to 6.7 per cent in 1928, and rose again to 7.3 per cent in 1929.

North Central District

Only 6.4 per cent of the total area of street pavement laid in the 201 cities was put down in the North-Central district, which lies west of Lake Michigan and the Mississippi River, north of St. Louis, Kansas City, Wichita, and Amarillo, and east of the Rocky Mountains. Although this district comprises 22.2 per cent of the area of continental United States, it contained in 1930 only 8,7 per cent of the total population. The economic basis of this broad area lies almost entirely in grain-farming, stock raising, and dairying.

Street pavement laid in North-Central district, 1925-1929, by types of paving
(In square yards)

Type	1925	1926	1927	1928	1929
Sheet asphalt	639,058	585,026	374,612	439,726	437,831
Asphaltic concrete	1,030,836	695,946	720,729	401,633	352,941
Asphaltic macadam	148,446	209,052	240,092	185,589	101,566
Asphalt block	--	--	--	--	--
Natural rock asphalt	--	--	--	--	--
Other asphaltic types of paving	--	--	--	--	1,783
Total bituminous	1,818,340	1,490,024	1,335,433	1,026,948	894,121
Portland-cement concrete .	1,910,141	1,847,964	2,056,513	1,571,858	1,621,428
Brick, block, and stone ..	125,377	133,041	182,722	193,720	240,081
Total	3,853,858	3,471,029	3,574,668	2,792,526	2,755,630

The total area of street paving laid in the North-Central district declined 28.5 per cent, from 3,853,858 square yards in 1925 to 2,755,630 square yards in 1929. More than one-third (35.7 per cent) was laid in Wisconsin, 22.7 per cent in Iowa, 20.9 per cent in Nebraska, and 18.1 per cent in Minnesota. Street pavement laid in Nebraska cities decreased 83.9 per cent, from 1,342,528 square yards in 1925 to 214,924 square yards in 1929; and in Iowa cities, 49.1 per cent, from 1,017,811 square yards in 1925 to 517,849 square yards in 1929. On the other hand, street pavement laid in Wisconsin increased 63.4 per cent, from 792,107 square yards in 1925 to 1,294,116 square yards in 1929.

The laying of bituminous pavements in the North-Central district decreased considerably from 1925 to 1929 both in yardage and in relative importance. In yardage it decreased 50.8 per cent, from 1,818,340 square yards in 1925 to 894,121 square yards in 1929. In their ratio to the total pavement laid, bituminous pavements declined from 47.2 per cent in 1925 to 42.9 per cent in 1926, 37.4 per cent in 1927, 36.8 per cent in 1928, and 32.4 per cent in 1929. Nearly half (48.8 per cent) of the bituminous pavement consisted of asphaltic concrete, 37.7 per cent of sheet asphalt, and 13.5 per cent of asphaltic macadam. The chief decrease occurred in asphaltic concrete, of which the yardage laid in 1929 was only 34.2 per cent of that laid in 1925. Sheet asphalt fell off 31.5 per cent during the same period and asphaltic macadam fell 31.6 per cent.

At the same time, the laying of Portland-cement concrete in the North-Central district decreased in yardage in 1928 and 1929, but increased continuously in relative importance. From 1,910,141 square yards in 1925 the total yardage of Portland-cement concrete increased to 2,056,513 square yards in 1927, but dropped to 1,571,858 square yards in 1928 and 1,621,428 square yards in 1929. The ratio of Portland-cement concrete pavement to the total pavement, however, increased from 49.6 per cent in 1925 to 53.2 per cent in 1926, 57.5 per cent in 1927, 56.3 per cent in 1928, and 58.9 per cent in 1929.

Nearly twice as great an area was paved with brick, block, or stone in the North-Central district in 1929 as in 1925. From 125,377 square yards in 1925 the yardage of brick, block, and stone pavements increased to 240,081 square

yards in 1929. By far the greater part of this yardage consisted of vitrified brick. At the same time the ratio of brick, block, and stone pavements to the total pavement laid increased from 3.2 per cent in 1925 to 3.8 per cent in 1926, 5.1 per cent in 1927, 6.9 per cent in 1928, and 8.7 per cent in 1929.

Pacific-Rocky Mountain District

One-fifth of all hard-surfaced pavements laid in the 201 cities from 1925 to 1929 were put down in the Pacific-Rocky Mountain district, which lies west of Great Falls, Cheyenne, Denver, Albuquerque, and El Paso. The 22 cities of this district from which data were received contained 10.2 per cent of the total population covered by this survey. Included in the 22 cities were one Class A city of over 1,000,000 population, one Class B city with a population between 500,000 and 1,000,000, four Class C cities with from 250,000 to 500,000 inhabitants, five Class D cities with 100,000 to 250,000 residents, and 11 Class E cities containing less than 100,000 persons. Although the Pacific-Rocky Mountain district comprised 34.4 per cent of the total area of continental United States, it contained in 1930 only 9.7 per cent of the total population and in 1930 only 9.7 per cent of the total population living in communities of more than 40,000. The basic industries of this district are chiefly agricultural and mining.

Street pavement laid in Pacific-Rocky Mountain district, 1925-1929, by types of paving
(In square yards)

Type	1925	1926	1927	1928	1929
Sheet asphalt..........	1,290,833	1,375,977	1,496,066	1,287,432	1,854,291
Asphaltic concrete	3,138,667	5,276,585	2,705,159	3,716,472	3,065,908
Asphaltic macadam	--	--	--	--	587
Asphalt block	--	--	--	--	--
Other asphaltic types of paving	765,357	398,445	581,203	305,235	413,179
Total bituminous	5,194,857	7,051,007	4,782,428	5,309,139	5,333,965
Portland-cement concrete	2,748,624	4,150,235	6,306,128	7,456,723	5,254,890
Brick, block, and stone	14,661	1,865	3,023	953	3,604
Total	7,958,142	11,203,107	11,091,579	12,766,815	10,592,459

The total area of street pavement laid in the Pacific-Rocky Mountain district increased 40.8 per cent, from 7,958,142 square yards in 1925 to 11,203,107 square yards in 1926, remained virtually stationary at 11,091,579 square yards in 1927, increased 15.1 per cent to 12,766,815 square yards in 1928, and dropped 17.0 per cent to 10,592,459 square yards in 1929. Ninety-five per cent of the total was laid in the three Pacific States, and 77.4 per cent in California alone.

The total yardage of bituminous pavement laid in the Pacific-Rocky Mountain district increased 35.7 per cent, from 5,194,857 square yards in 1925 to 7,051,007 square yards in 1926, but dropped 32.2 per cent to 4,782,428 square

yards in 1927, increasing 11.0 per cent to 5,309,139 square yards in 1928 and remaining practically level at 5,333,965 square yards in 1929. The ratio of bituminous pavements to the entire area of street pavement decreased from 65.3 per cent in 1925 to 62.9 per cent in 1926, dropped to 43.1 per cent in 1927 and to 41.6 per cent in 1928, and rose to 50.4 per cent in 1929.

Sixty-five per cent of all bituminous surfaces laid in the Pacific-Rocky Mountain district consisted of asphaltic concrete of various types. Sheet asphalt formed only 26.4 per cent of the whole; and miscellaneous bituminous types, only 8.9 per cent.

The use of Portland-cement concrete for street paving in the Pacific Rocky Mountain district reached its peak in 1928 with 7,456,723 square yards, an increase of 71.3 per cent over the 2,748,624 square yards laid in 1925. In 1929, however, only 5,254,890 square yards of Portland-cement concrete was laid, a decrease of 29.5 per cent from the yardage laid in 1928 but an increase of 91.2 per cent over that laid in 1925. The ratio of Portland-cement concrete to the total pavement increased from 34.6 per cent in 1925 to 37.0 per cent in 1926, 56.8 per cent in 1927, and 58.5 per cent in 1928, but dropped to 49.5 per cent in 1929.

Ninety-eight per cent of the Portland-cement concrete was laid in the three Pacific States. Seven-tenths of the total was laid in California alone.

DISCUSSION BY POPULATION GROUPS

Class A Cities

In the five Class A cities with populations exceeding 1,000,000, the total yardage of new pavement and resurfacing laid increased 44.1 per cent from 13,083,312 square yards in 1925 to 18,856,836 square yards in 1927. The yardage laid in 1928 was nearly equal to that laid in 1927, but the 17,315,158 square yards laid in 1929 was 7.5 per cent less than the 18,714,749 square yards laid in 1928.

Marked divergences from the general trend of street paving in Class A cities from 1925 to 1929 were shown by the various types of street paving. Bituminous pavements in general increased 23.5 per cent in area from 1925 to 1927, decreased 1.5 per cent in 1928, and rose 2.1 per cent in 1929, to a figure 24.2 per cent higher than in 1925. Their share in the total pavement dropped from 73.9 per cent in 1925 and 72.1 per cent in 1926 to 63.3 per cent in 1927 and 62.8 per cent in 1928, but rose to 69.3 per cent in 1929. The area of Portland-cement pavements increased 170 per cent from 1925 to 1927, and 11.7 per cent in 1928, but dropped 21.8 per cent in 1929. As a result, the ratio of Portland-cement concrete to the total pavement increased from 14.3 per cent in 1925 to 17.5 per cent in 1926, 26.8 per cent in 1927, and 30.2 per cent in 1928, but dropped to 25.5 per cent in 1929. Brick, block, and stone pavements showed a smaller proportionate increase up to their peak in 1927, and a greater proportionate decrease. Their share in the total pavement decreased from 11.8 per cent in 1925 and 10.4 per cent in 1926 to 9.8 per cent in 1927, 7.0 per cent in 1928, and 5.1 per cent in 1929.

Street pavement laid in Class A cities, 1925-1929, by types of paving
(In square yards)

Type	1925	1926	1927	1928	1929
Sheet asphalt	8,636,971	9,724,850	10,436,770	9,598,608	9,634,009
Asphaltic concrete	840,157	1,430,852	1,159,674	1,771,392	1,496,198
Asphaltic macadam	126,851	190,827	85,462	133,597	503,789
Asphalt block	66,480	60,726	116,108	63,039	46,605
Natural rock asphalt	---	---	31,724	49,918	---
Other asphaltic types of paving	---	49,903	111,581	144,376	327,277
Total bituminous	9,670,459	11,457,158	11,941,319	11,760,930	12,007,878
Portland-cement concrete	1,873,737	2,776,423	5,060,305	5,650,751	4,421,605
Brick, block, and stone .	1,539,116	1,651,378	1,855,122	1,303,068	885,675
Total	13,083,312	15,884,959	18,856,836	18,714,749	17,315,158

Class B Cities

In the eight Class B cities with populations ranging from 500,000 to 1,000,000,--Baltimore, Boston, Buffalo, Cleveland, Milwaukee, Pittsburgh, St. Louis, and San Francisco--the general trend of paving was similar to that in the Class A cities. It rose 30.4 per cent from 4,083,575 square yards in 1925 to a peak of 5,325,020 square yards in 1927, fell off 4.0 per cent to 5,111,777 square yards in 1928, and 2.0 per cent to 5,011,890 square yards in 1929.

Bituminous pavements followed somewhat the general trend, increasing 21.6 per cent in area from 1925 to 1927, decreasing 1.8 per cent in 1928 and 2.3 per cent in 1929. As a result their share of the total pavement laid increased from 66.6 per cent in 1925 to 67.5 per cent in 1926, dropped to 62.5 per cent in 1927, rose to 64.0 per cent in 1928, and decreased to 63.8 per cent in 1929. The yardage of Portland-cement concrete doubled from 1925 to 1927, increased 1.2 per cent in 1928, but decreased 10.6 per cent in 1929. Its share in the total paving increased from 14.8 per cent in 1925 to 20.4 per cent in 1926, 23.0 per cent in 1927, and 24.3 per cent in 1928, but fell off to 22.6 per cent in 1929. The trend of brick, block, and stone paving in Class B cities from 1925 to 1929 was quite irregular. The yardage laid in 1929 was 10.2 per cent lower than in 1925 and the relative importance had dropped from 18.6 per cent of the total in 1925 to 13.6 per cent in 1929.

Street pavement laid in Class B cities, 1925-1929, by types of paving
(In square yards)

Type	1925	1926	1927	1928	1929
Sheet asphalt	1,734,716	2,038,197	2,294,740	2,192,673	2,224,408
Asphaltic concrete	859,463	956,510	886,133	884,842	885,421
Asphaltic macadam	122,385	186,501	149,814	193,495	88,362
Asphalt block	--	--	--	--	--
Natural rock asphalt ...	4,570	3,690	--	--	--
Other asphaltic types of paving	--	--	--	--	--
Total bituminous	2,721,134	3,184,898	3,330,687	3,271,010	3,198,191
Portland-cement concrete	604,374	963,141	1,223,780	1,243,799	1,132,310
Brick, block, and stone	758,067	569,902	770,553	596,968	681.389
Total	4,083,575	4,717,941	5,325,020	5,111,777	5,011,890

Class C Cities

Less proportionate change occurred in the total pavement laid in Class C cities[6] from 1925 to 1929 than in Class A or Class B cities. The total yardage laid decreased 3.3 per cent from 8,522,775 square yards in 1925 to 8,263,222 square yards in 1926; increased 10.2 per cent to a maximum of 9,104,521 square yards in 1927, decreased 7.3 per cent to 8,484,832 square yards in 1928, and 2.6 per cent to 8,296,574 square yards in 1929, 2.7 per cent less than the yardage laid in 1925. Bituminous pavements decreased about 7.5 per cent a year from 1925 to 1928 but increased 4 per cent in 1929. Their share in the total pavement laid decreased from 67.4 per cent in 1925 and 62.6 per cent in 1926 to 53.7 per cent in 1927 and 52.4 per cent in 1928, but rose to 55.7 per cent in 1929. On the other hand, Portland-cement concrete showed a sharp rise of 52.5 per cent from 1925 to 1927, followed by a decline of 1.2 per cent in 1928 and of 14.4 per cent in 1929. The share of Portland-cement concrete in the total pavement increased from 26.4 per cent in 1925 to 30.5 per cent in 1926, 37.6 per cent in 1927, and 39.9 per cent in 1928, but decreased 35.7 per cent in 1929. Brick, block, and stone pavements rose irregularly from 1925 to 1927, dropped in 1928, and rose in 1929. Their share in the total pavement rose from 6.2 per cent in 1925 to 7.0 per cent in 1926, 8.7 per cent in 1927, 7.7 per cent in 1928, and 8.6 per cent in 1929.

Street pavement laid in Class C cities, 1925-1929, by types of paving
(In square yards)

Type	1925	1926	1927	1928	1929
Sheet asphalt	2,492,968	1,965,890	2,131,229	2,243,750	1,892,238
Asphaltic concrete	2,591,872	2,717,049	2,052,788	1,545,745	1,837,570
Asphaltic macadam	2,003	--	--	5,931	--
Asphalt block	--	--	--	--	--
Natural rock asphalt	591,754	427,884	540,141	505,789	796,924
Other asphaltic types of paving	66,588	59,353	160,876	141,034	92,000
Total bituminous	5,745,185	5,170,176	4,885,034	4,442,249	4,618,732
Portland-cement concrete	2,249,853	2,518,514	3,426,345	3,385,838	2,959,896
Brick, block, and stone..	527,737	574,532	792,642	656,745	717,946
Total	8,522,775	8,263,222	9,104,521	8,484,832	8,296,574

6/ The following cities, with populations ranging between 250,000 and 500,000, have been included in Class C: Akron, O.; Atlanta, Ga.; Birmingham, Ala.; Cincinnati, O.; Columbus, O.; Dallas, Tex.; Denver, Colo.; Houston, Tex.; Indianapolis, Ind.; Jersey City, N. J.; Kansas City, Mo.; Louisville, Ky.; Memphis, Tenn.; Minneapolis, Minn.; Newark, N. J.; New Orleans, La.; Oakland, Calif.; Portland, Oreg.; Providence, R. I.; Rochester, N. Y.; San Antonio, Tex.; Seattle, Wash.; Toledo, O.; and Washington, D. C.

Class D Cities

The total area of street pavement laid in Class D cities[7] increased 14.0 per cent, from 8,244,707 square yards in 1925 to 9,394,458 square yards in 1926. From this peak the area paved each year decreased until in 1929 only 7,912,153 square yards was laid, 4.2 per cent less than in 1925. Bituminous pavements in general shared in the increase in 1926 and in the decline from 1927 to 1929. Their ratio to the total pavement decreased from an average of 66 per cent between 1925 and 1927 to 63.6 per cent in 1928 and 60 per cent in 1929. Decreases in the yardage of sheet asphalt and of asphaltic concrete laid were largely compensated by increased yardage of asphaltic macadam, asphalt block, and natural rock asphalt. On the other hand, Portland-cement concrete increased steadily both in yardage and in importance. Its share in the total pavement increased from 25.2 per cent in 1925 and 1926 to 29.1 per cent in 1927, 30.8 per cent in 1928, and 33.8 per cent in 1929. Brick, block, and stone pavements, after a sharp rise in 1926, dropped in 1927, 1928, and 1929 to a level 12 per cent lower than in 1925. Their share in the total paving rose from 6.6 per cent in 1925 to 10.9 per cent in 1926, dropped to 5.4 per cent in 1927, and 5.6 per cent in 1928, and rose again to 6.2 per cent in 1929.

Street pavement laid in Class D cities, 1925-1929, by types of paving
(In square yards)

Type	1925	1926	1927	1928	1929
Sheet asphalt	2,271,972	2,611,297	2,839,013	2,270,972	1,672,470
Asphalt concrete	2,614,112	2,403,611	2,375,969	2,476,083	1,856,345
Asphaltic macadam	356,142	475,702	278,859	549,633	868,316
Asphalt block	165,833	325,796	191,686	2,272	55,909
Natural rock asphalt	79,096	10,484	118,210	197,794	273,943
Other asphaltic types of paving	134,133	278,373	42,177	8,009	18,385
Total bituminous	5,621,288	6,005,263	5,845,914	5,504,763	4,745,362
Portland-cement concrete	2,074,780	2,371,868	2,594,545	2,679,039	2,674,530
Brick, block, and stone..	549,639	1,017,327	487,495	488,518	492,261
Total	8,244,707	9,394,458	8,927,954	8,672,320	7,912,153

7/ The following cities, with populations ranging between 100,000 and 250,000, have been included in Class D: Albany, N. Y.; Birmingham, Ala.; Bridgeport, Conn.; Cambridge, Mass.; Camden, N. J.; Canton, O.; Chattanooga, Tenn.; Dayton, O.; Des Moines, Iowa; Duluth, Minn.; El Paso, Tex.; Elizabeth, N. J.; Fall River, Mass.; Flint, Mich.; Fort Wayne, Ind.; Fort Worth, Tex.; Grand Rapids, Mich.; Hamtramck, Mich.; Hartford, Conn.; Jacksonville, Fla.; Kansas City, Kans.; Knoxville, Tenn.; Long Beach, Calif.; Lowell, Mass.; Lynn, Mass.; Miami, Fla.; Nashville, Tenn.; New Bedford, Mass.; New Haven, Conn.; Norfolk, Va.; Oklahoma City, Okla.; Omaha, Nebr.; Richmond, Va.; St. Paul, Minn.; Salt Lake City, Utah; San Diego, Calif.; Savannah, Ga.; Scranton, Pa.; Somerville, Mass.; Spokane, Wash.; Syracuse, N. Y.; Tacoma, Wash.; Tampa, Fla.; Trenton, N. J.; Tulsa, Okla.; Utica, N. Y.; Wichita, Kans.; Wilmington, Del.; Worcester, Mass.; Yonkers, N. Y.; Youngstown, O.

Class E Cities

The area of pavement laid in Class E cities 8/ increased 17.8 per cent, from 13,352,216 square yards in 1925 to 15,729,920 square yards in 1927, but decreased 21.1 per cent to 12,511,247 square yards in 1928 and 21.1 per cent again to 10,332,261 square yards in 1929. Bituminous pavements followed the general trend, reaching their peak, however, in 1926 instead of 1927. Their ratio to the total pavement decreased from 70.4 per cent in 1925 and 70.0 per cent in 1926 to 65.7 per cent in 1927 and 65.6 per cent in 1928, but rose to 68.8 per cent in 1929. The area paved with Portland-cement concrete rose sharply from 1925 to 1927, but dropped almost as sharply in 1928 and 1929. Its ratio to the total pavement increased steadily from 16.8 per cent in 1925 and 18.2 per cent in 1926 to 22.9 per cent in 1927, 23.4 per cent in 1928, and 25.1 per cent in 1929. Brick, block, and stone pavement, after a relatively small increase in 1926 and 1927, dropped sharply in 1928 and 1929. Its share in the total paving decreased from 12.8 per cent in 1925 to 11.8 per cent in 1926, 11.4 per cent in 1927, 10.9 per cent in 1928, and 6.1 per cent in 1929.

Street pavement laid in Class E cities, 1925-1929, by types of paving
(In square yards)

Type	1925	1926	1927	1928	1929
Sheet asphalt	3,639,384	3,249,139	3,703,120	2,829,122	2,508,970
Asphaltic concrete	2,939,538	4,680,678	3,284,716	3,051,624	2,153,043
Asphaltic macadam	739,554	649,678	855,701	836,663	1,126,353
Asphalt block	184,210	574,973	683,298	138,676	31,280
Natural rock asphalt ...	177,305	372,386	473,260	346,439	15,913
Other asphaltic types of paving	1,599,861	1,132,183	1,301,055	1,002,051	1,262,214
Tar macadam	114,433	39,055	27,917	10,099	10,000
Total bituminous	9,394,285	10,698,092	10,329,067	8,214,674	7,107,773
Portland-cement concrete	2,247,310	2,777,905	3,603,392	2,929,717	2,588,740
Brick, block, and stone	1,710,621	1,802,183	1,797,461	1,366,856	635,748
Total	13,352,216	15,278,180	15,729,920	12,511,247	10,332,261

8/ The following cities, with populations ranging for the most part between 40,000 and 100,000, are included in Class E: Allentown, Pa.; Altoona, Pa.; Atlantic City, N. J.; Augusta, Ga.; Aurora, Ill.; Austin, Tex.; Battle Creek, Mich.; Bayonne, N. J.; Beaumont, Tex.; Berkeley, Calif.; Bethlehem, Pa.; Binghamton, N. Y.; Boise, Idaho; Brockton, Mass.; Brookline, Mass.; Burlington, Vt Butte, Mont.; Cedar Rapids, Iowa; Charleston, S. C.; Charleston, W. Va.; Charlotte, N. C.; Chester, Pa.; Chicopee, Mass.; Cicero, Ill.; Columbia, S. C.; Columbus, Ga.; Council Bluffs, Iowa; Davenport, Iowa; Decatur, Ill.; Dubuque, Iowa; Durham, N. C.; East Chicago, Ind.; East Orange, N. J.; East Saint Louis, Ill.; Elmira, N. Y.; Erie, Pa.; Evanston, Ill.; Evansville, Ind.; Everett, Mass.; Fargo, N. D.; Fitchburg, Mass.; Fresno, Calif.; Galveston, Tex.; Gary, Ind.; Hamilton, Ohio; Hammond, Ind.; Harrisburg, Pa.; Haverhill, Mass.; Highland Park, Mich.; Holyoke, Mass.; Huntington, W. Va.; Jackson, Mich.; Jackson, Miss.; Jamestown, N. Y.; Johnstown, Pa.; Joliet, Ill.; Kalamazoo, Mich.; Kenosha, Wis.; Kokomo, Ind.; Lakewood, Ohio; Lancaster, Pa.; Lansing, Mich.; Lawrence, Mass.; Lincoln, Nebr.; Lima, Ohio; Lorain, Ohio; Macon, Ga.; Madison Wis.; Manchester, N. H.; Mobile, Ala.; Muskegon, Mich.; New Castle, Pa.; New Britain, Conn.; New Brunswick, N. J.; Newton, Mass.; Niagara Falls, N. Y.; Oak Park, Ill.; Pasadena, Calif.; Passaic, N. J.; Pawtucket, R. I.; Phoenix, Ariz.; Pontiac, Mich.; Portsmouth, Ohio; Portsmouth, Va.; Portland, Me.; Pueblo, Colo. Racine, Wis.; Reno, Nev.; Rockford, Ill.; Sacramento, Calif.; Saginaw, Mich.; San Jose, Calif.; St. Joseph, Mo.; St. Petersburg, Fla.; Schenectady, N. Y.; Sioux City, Iowa, Sioux Falls, S. Dak.; South Bend, Ind.; Springfield, Ill.; Springfield, Ohio, Stockton, Calif.; Superior, Wis.; Terre Haute, Ind.; Topeka Kans.; Waco, Tex.; Waterbury, Conn.; Wichita Falls, Tex.; Williamsport, Pa.; Wheeling, W. Va.; Winston-Salem, N. C.; Wilkes-Barre, Pa.; Woonsocket, R. I.; York, Pa.

INFORMATION CIRCULAR

DEPARTMEN OF COMMERCE - BUREAU OF MINES

SPECIALLY RECOMMENDED TRAILING CABLES[1]

(Cables added to recommended list during December, 1930)

By L. C. Ilsley[2]

A specially recommended cable is a rubber-sheathed cable that has met the minimum performance requirements for cables as set forth in Schedule 2C.[3]

The first in the series of recommended cables was listed September, 1930, by an advance notice published as Information Circular 6375[4] which described the conditions under which the cable tests were made. Since that time six additional cables have met the test requirements and have been specially recommended. The complete list of specially recommended. cables is as follows:

Specially recommended=cables

Cable	Symbol No.	Stranding	Manufacturer
Hazacord No. 2	BM1	19 x 7	Hazard Insulated Wire Works (division of the Okonite Co.)
Hazacord No. 3	BM2	do.	Do.
Hazacord No. 4	BM3	do.	Do.
Super-service No. 3	BM4	7 x 19	Rome Wire Company (division of the General Cable Corporation)
Super-service No. 2	BM5	do.	Do.
Super-service No.. 4	BM6	do.	Do.
Super-service No. 4	BM7	7x 7	Do.

The cables listed were all twin (parallel duplex) cables having outer belts or coverings of approximately ⅛-inch high-grade rubber compound, and have the same general appearance and relative dimensions as the types of twin cables in general use.

The cables were tested under conditions designed to show their approximate performance under mine conditions. Only twin-type cables have met these test conditions. Although the recommended trailing cables are much better and safer for mine use than other cables and withstand being run over by a 7-ton car, they should not be considered as 100 per cent safe.

The 7-ton car test represents the minimum limit requirement effective at present under Schedule 2-C, but the test car operates on a straight and level track, and thus does not represent the more severe conditions of mine use such as might result from uneven and curved tracks.

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6442."

2 Electrical engineer, U. S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

3 U. S. Bureau of Mines, Explosion-Proof Mine Equipment Requirements for Approval of Storage-Battery Locomotives and Power Trucks, Junction Boxes and Electric Motor-Driven Equipment: Schedule 2C, February, 1930, 27 pp.

4 Ilsley, L. C., Special Recommonded Trailing Cable (advance notice): Information Circular 6375, Bureau of Mines.

4 Ilsley, L. C., Special Recommended Trailing Cable (advance notice): I. C. 6375, Bureau of Mines, 1930, 2pp.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

REQUIREMENTS FOR BOLTS AND SIMILAR FASTENINGS FOR PERMISSIBLE ELECTRICAL EQUIPMENT[1]

By L. C. Ilsley[2]

The observations made by Bureau of Mines engineers of permissible equipment installed in about 40 mines have indicated a chance for improvement in designs calling for the use of bolts, screws, studs, etc., especially with respect to bolt sizes and protection from mechanical injury.

Minimum Size of Bolts

In a recent conference with engineers of a large manufacturing company the question of bolt sizes was being discussed, and a story was told which fits the case so well that it is included here. It seems that a young engineer brought a design having $\frac{1}{4}$-inch bolts to an old-time engineer for criticism. The latter said the bolts should be larger; the other argued that they had an adequate factor of safety. The old engineer said "Young man, in mining work engineering sense begins with $\frac{1}{2}$-inch bolts." And this is the position Bureau of Mines engineers take after seeing the trouble experienced in maintaining equipment in mines where smaller than $\frac{1}{2}$-inch bolts are used. Having been convinced of this point, the bureau is urging everyone to take this matter under careful consideration when bringing out any new designs.

Protection of Bolt Heads

Field investigations have also shown that where bolt heads are exposed to the wear of chains, ropes, or even coal and rock ribs, the equipment is rendered less than safe and in effect becomes substandard. This point should be kept in mind when laying out new designs where permissibility is involved.

Spacing of Bolts

No fixed rule for spacing bolts can be laid down. Compartments having flanges in one plane will require closer bolting space than stepped flanges. The bureau's criterion of sufficient bolting rests solely on whether flames are restricted. It is not good practice to have a bolt so located with respect to a corner as to leave the corner to any considerable extent unclamped in case

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6432."

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the corner bolt is missing. As far as feasible, to be effective bolts should be on or near the flange center line. A large number of bolts prevent ready access to the compartments. This fault can in some cases be largely overcome by providing more accessible inspection openings in the bolted cover.

Projecting Studs to be Avoided

Studs should not project through the nuts. In a typical case the thread which is above the nut becomes battered, and in trying to remove the nut the stud is loosened. Each time the stud is removed, dirt gets into the hole, and finally the stud enters the hole only a little way, stripping the few effective threads. This means that the hole must be re-drilled and tapped for a larger stud, also the hole in the cover must be enlarged and an oversize stud obtained. In many cases the operator does not go to this trouble and no fastening is used at that particular point. This is an important point in the maintenance of permissible outfits.

Length of Cap Screws and Studs

The bureau is checking the size of cap screws and studs used in bottomed holes to make sure that they are not too long. If they are too long they may bear on the bottomed hole before the head has a chance to do its work in tightening the compartment. Cap screws and studs should not bottom even if the lock washer is omitted. Flat-headed screws are particularly difficult to inspect for this fault.

Securing Nuts

Various schemes are used by designersto keep nuts from loosening. Lock washers, castellated nuts, and nuts with drilled holes are some of the devices used. Where a number of nuts are wired together, it is difficult to save the wire, consequently unless a supply of wire is kept on hand it is liable to be left off when the repair job is completed. In a good many cases, small iron wire is used which rapidly deteriorates. Wires are also likely to catch on clothes or scratch the operator's hands. A larger nonrusting seal wire would help in making the wires last longer. In summing up the situation, it may be said that the lock washer under the nut seems to be most suitable for a majority of cases.

I. C. 6433

March, 1931

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

AMALGAMATION PRACTICE AT PORCUPINE UNITED GOLD MINES, LTD., TIMMINS, ONT.

BY

Ronald A. Vary

I.C.6433,
March, 1931.

INFCRMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

AMALGAMATION PRACTICE AT PORCUPINE UNITED GOLD MINES, LTD., TIMMINS, ONT.[1]

By Ronald A. Vary[2]

INTRODUCTION

This paper is one of a series prepared by the United States Bureau of Mines dealing with milling methods and costs. In view of the current revival of interest in gold mining, information dealing with practices and costs in the development of gold properties is apropos. Milling practice at a small gold property in the prospect stage is described where, with a small outlay for plant and equipment, development has been carried on for about 20 months on the proceeds of the mill operation with little additional expenditure.

The paper shows about what may be expected as to costs at a small property operated under northern climatic conditions and with rigid economy throughout the plant.

GENERAL DESCRIPTION

The Porcupine United Gold Mines is a reorganization of a group of claims formerly owned by Canadel Gold, Ltd. A considerable body of ore was partly developed by the former company, but work was stopped for four years or more until the property was taken up by the present company. The main claims where development work is being carried on adjoin the property of the Hollinger Gold Mines and McIntyre Gold Mines on their north boundary.

The ore which occurs in narrow but high-grade lenses is mined by shrinkage stoping. It consists of quartz with fine stringers of mineralized basaltic schist, or of banded quartz schist, or of stringers of quartz in schist, accompanied by pyrite and often containing visible gold. The gangue rock is basaltic schist.

The mill is an amalgamation plant of 25 tons daily capacity, installed as a pilot mill to help defray costs of development and to treat high-grade ore, with a cyanide plant the final objective as soon as mining developments warrant one.

Water for the mill is obtained from the mine, the underground pumps discharging directly to the mill tank.

Power is furnished by the Northern Canada Power Co. which supplies all the mines of northern Ontario. The power is transmitted from hydroelectric plants at 11,000 volts and is stepped down at the property to a 550-volt, 3-phase, 25-cycle current which operates all mine motors.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used "Reprinted from U. S. Bureau of Mines Information Circular 6433."

2 - One of the consulting engineers, U. S. Bureau of Mines, and mill foreman, Porcupine United Gold Mines, Ltd.

About 75 per cent of the gold in the ore is free milling and is mostly fairly coarse.

The plant was put in operation on March 24, 1929, and has treated 20 to 30 tons of ore daily up to the present.

CRUSHING

Hoisting and crushing are done on day shift only. Large pieces of barren rock are sorted from the ore underground and disposed of as waste. The milling ore is trammed to the shaft and hoisted to the surface in cars of approximately 1-ton capacity. The crushing plant is located in the shaft house and the ore is dumped as hoisted on a grizzly of 40-pound rails spaced 5 inches apart, which covers the whole of the crusher ore bin. Large chunks are broken up and put through the grizzly by the shaft deck man. The capacity of the crusher bin is 15 tons. The crude ore is fed to a 7 by 10-foot Allis-Chalmers jaw crusher driven by a 15-hp. motor. The crusher breaks the ore to 1 by 1-1/2 inches and discharges it to a 12-inch conveyor belt, feeding a 10 by 18 inch Allis Chalmers rolls (see flow sheet, fig. 1). The product from the rolls, 75 per cent of which is less than 1/2-inch mesh, falls on a 12-inch conveyor belt, traveling 100 feet per minute, and is conveyed to the mill which is located 75 feet from the shaft house. A 10-hp. motor belted to a countershaft drives the rolls and the roll-feeder conveyor. Both rolls are operated at 90 r.p.m. Occasional burning off of high ridges on the shells is required to keep them in shape. The crusher plant is operated by one man who does the necessary repair work and helps in the mill when he is not tending the crusher. The 12-inch conveyor discharges into the boot of an elevator with 10-inch buckets spaced 18 inches apart, 33-foot centers, speed 250 feet per minute, which delivers the ore to the mill bin. This bin is 12 by 12 by 10 feet and is constructed of 3-inch planks. It has a capacity of about 50 tons.

GRINDING

Grinding is done in a 4-1/2-foot by 16-inch Hardinge mill, carrying a ball load of approximately 4,000 pounds. It is operated at 31 r.p.m. and grinds an average of 25 tons of ore per 24 hours to 65 per cent through 200 mesh. A circulating load of 125 per cent is maintained in the ball mill. The discharge from the ball mill contains 32 to 36 per cent solids. Four-inch forged-steel balls are used. Their wear is approximately 1-1/2 pounds per ton of ore ground. The standard, wedge-bar, manganese-steel liners in the mill last 12 to 15 months. The ball mill is in closed circuit with two amalgamation plates and a 4 by 15 foot Duplex Dorr classifier inclined 2 inches to 1 foot and making 26 strokes per minute. The classifier rake product is returned by gravity to the ball-mill feeder scoop where water is added for regulation of the moisture content of the ball-mill charge. The overflow of the classifier is sampled hourly by hand by taking dip samples from its overflow launder. The ball-mill is belt-driven from a line shaft which is belted to a 50-hp. motor. The line shaft and motor supply power for all mill machinery including the 12-inch conveyor belt from the crusher plant and the elevator. The Dorr classifier is driven by a 5-hp., 3-phase, 550-volt motor.

AMALGAMATION

The ball-mill discharge falls into a distributing box where water is added and the flow distributed to two 4 by 8 foot copper amalgamation plates 1/8-inch thick and with a slope of 1-5/8 inches to 1 foot. The distributing box traps considerable coarse gold. The

box is cleaned out once a week and its contents are run through an amalgamation barrel. The table frames supporting the amalgamation plates are constructed of 2 by 4 inch lumber, with cross pieces placed on edge and spaced 6 inches apart. The table decks are made of 1-inch shiplap on which the copper plates are screwed. The slope of the deck can be changed to suit conditions by placing wedges between the deck and the table frame. The plates are not silvered and the copper has to be treated with a weak cyanide solution before the mercury will amalgamate with the copper. Little trouble has been experienced by copper showing on the plates.

Mercury is shaken on the top half of the plates and none is added elsewhere. The plates are ordinarily dressed every three hours, but oftener if the ore is rich, depending upon the judgment of the operator. The method employed in cleaning a plate is to by-pass all the feed to the other plate, clean off all ore particles, then brush the plate well with a stiff whisk broom, working any loose amalgam to the top of the plate. This loosened amalgam is removed and if this leaves the plate too dry, mercury is shaken on and rubbed in well. The plate is then brushed horizontally, working from the center to the sides and starting at the bottom and working to the top of the plate. Any amalgam or loose mercury adhering to sides of the plate is then brushed to the top or removed if the amount is appreciable.

On the morning shift the plates are given an extra brushing and mercury is added to loosen the amalgam. Then the amalgam is stripped off with a piece of rubber conveyor belting, stripping being done horizontally to the flow on the plate; the amalgam is lifted and the plates are redressed in the usual manner. Care is taken that the plates are not stripped too clean.

Very little crystallization of the copper takes place, and the plates are rubbed occasionally with a weak cyanide solution which removes any tarnish or stains. Sufficient water is used to maintain an even flow of pulp over the plates and when the ball-mill discharges too much coarse material, the feed is cut off for a short period. Forty per cent of the total gold recovery is made on the amalgamation plates.

BLANKET TREATMENT

On the lower end of each table below the amalgamation plate, a sheet of 1/8-inch iron plate, 42 by 48 inches, is fastened. A blanket of No. 6 silence cloth is laid on this plate and secured by a flat iron bar 1/8-inch thick and 2 inches wide laid on top of the blanket. The bar is held by notches cut in the table frame. The pulp from the amalgamation plates passes over the blankets and pyrite, fine gold, and mercury from the plates are caught on the blankets. The blankets are changed and washed in a tub after each dressing of the plates, the blanket concentrates being sent to the amalgamation barrel for treatment. The plate and blanket tailings drop to amalgam traps at the end of each table and are elevated therefrom by an 8-inch bucket elevator which returns them to the Dorr classifier. This elevator has a deep sump which is a good trap and which is cleaned out at regular intervals. Amalgamation takes place in the whole circuit. Elevator discharge launders, classifier and other launders, all collect rich sands and amalgam; these are cleaned out periodically and treated in the amalgamation barrel. Thirty-five per cent of the gold is recovered by barrel treatment.

CONCENTRATION

The rake product from the Dorr classifier is returned to the ball mill, and the

overflow runs by gravity to a Gibson impact amalgamator, attached to a James sand concentrating table. The amalgamator catches float mercury and fine gold which has escaped the amalgamation plates, blankets, and traps. The amalgamator is opened, washed, and its plates are scraped once a week. The James table is operated at 250 r.p.m. with a 3/4-inch stroke. The table concentrates, averaging $40 per ton, are dewatered and sent for cyanide treatment to an affiliated company. The table tailings are elevated by an 8-inch bucket elevator to the tailings dump. The table concentrates and table tailings are both sampled hourly by hand by taking dip samples from their launders.

BARREL AMALGAMATION

The concentrates or sands from the blankets, traps, launders, etc., are ground for 10 hours in a cast-iron amalgamation barrel, 16 inches in diameter and 36 inches long, revolving at 22 r.p.m., using worn balls from the ball mill as grinding media. Then about 250 ounces of mercury and 3 pounds of slacked lime are added to the charge and it is again ground from 5 to 8 hours. The barrel is washed out into a box, the iron balls are carefully cleaned by hand, and the residue is run over a small amalgamation plate to the mill circuit. The mercury and amalgam are collected, washed, and cleaned with hot water and then squeezed by hand through fine sheeting to eliminate excess mercury, retaining the amalgam in the form of a ball.

The amalgam is retorted outside the mill over a wood fire at regular intervals, using a cast-iron retort which has a capacity of 1,000 ounces. The sponge gold recovery is 35 to 40 per cent of the weight of the amalgam retorted and the mercury loss is small. The sponge gold is melted in an oil-burning furnace at the affiliated company's refinery. Soda, borax, and manganese dioxide are used for flux, and the moulds are coated with lampblack. The bullion is sampled by drilling small holes in opposite ends of the bar at top and bottom. The average grade of bullion is: Gold, 770 fine; silver, 120 to 140 fine.

The crew for the mill consists of two amalgamators working eight hours each. On the day shift the mill is operated by a mill foreman who takes care of cleaning the amalgam, retorting, and melting.

On a per-ton basis, labor costs are as follows:

	Man-hours per ton
Crushing	0.32
Amalgamation	0.64
Plant foreman	0.32
Total labor	1.28

Table 1.—Assays

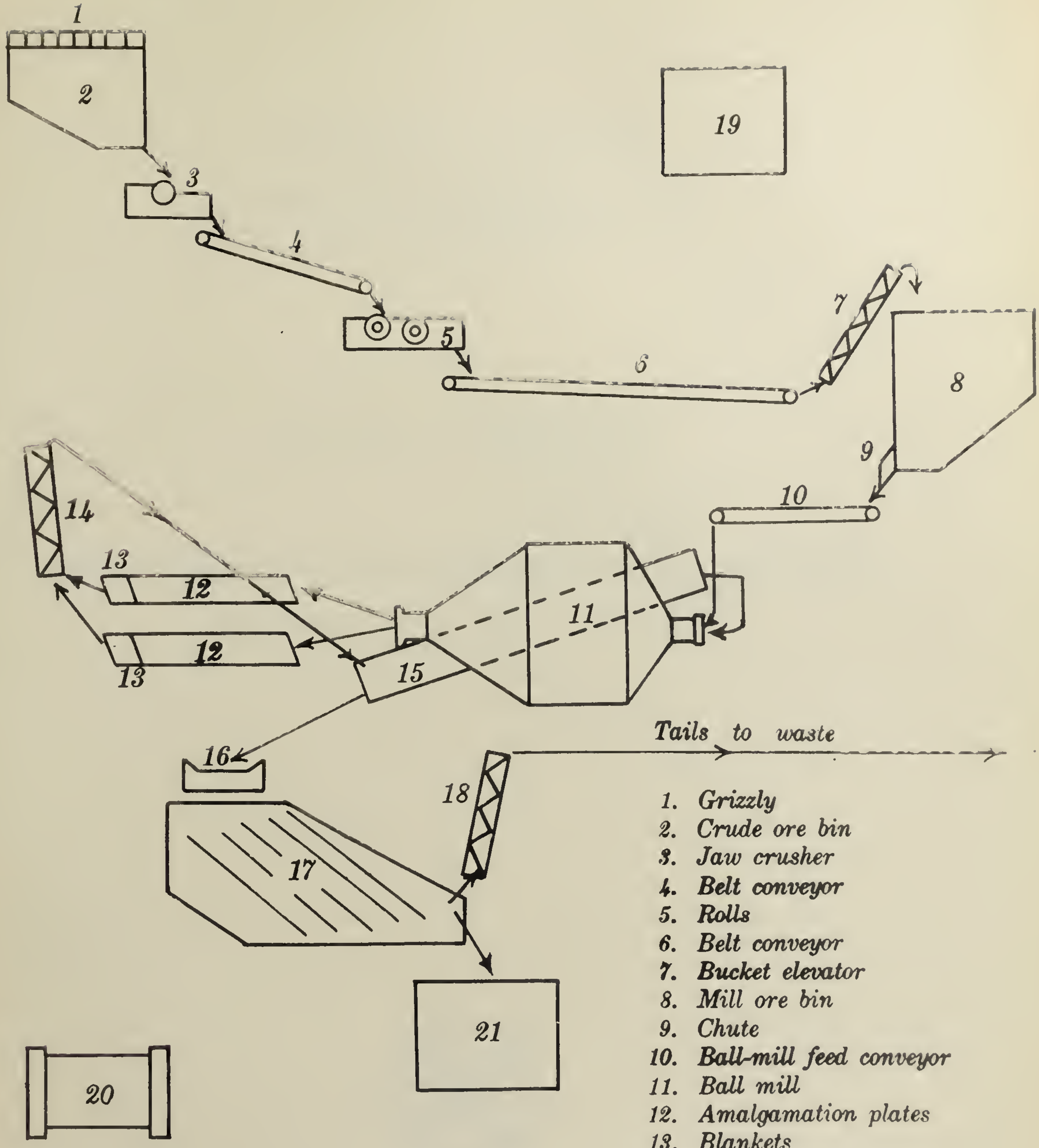
	Gold content per ton
Average heads to amalgamation plant	$11.00
Table heads (amalgamation, tailings)	2.80
Table tailings	1.80
Table concentrates	40.00

Table 2.--Operation costs per ton of ore treated

	Labor	Power	Supplies	Total
Crushing	$.214	$.135	$.035	$.384
Grinding	.429	.180	.162	.771
Classifying, screening, conveying, and refining	.429	.090	.107	.626
Miscellaneous			.070	.070
Totals	$1.072	$.405	$.374	$1.851

Table 3.--Unit supply consumption

1. Steel and Iron	
Ball consumption, pounds per ton ore milled	1.50
Liner consumption, pounds per ton ore milled	.48
2. Mercury, ounces per ton ore milled	.05
3. Power, kw.h. per ton ore milled	40.00



1. Grizzly
2. Crude ore bin
3. Jaw crusher
4. Belt conveyor
5. Rolls
6. Belt conveyor
7. Bucket elevator
8. Mill ore bin
9. Chute
10. Ball-mill feed conveyor
11. Ball mill
12. Amalgamation plates
13. Blankets
14. Bucket elevator
15. Dorr classifier
16. Gibson amalgamatior
17. James tables
18. Tailings elevator
19. Water tank
20. Amalgamation barrel
21. Concentrates bin

Figure 1. -- Flow sheet

I. C. 6434

April, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

SUPERVISION AS A MEANS OF PREVENTING ACCIDENTS FROM FALLS OF ROOF AND COAL

BY

W. H. FORBES

I.C.6434.
April, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SUPERVISION AS A MEANS OF PREVENTING ACCIDENTS FROM FALLS OF ROOF AND COAL[1]

By W. H. Forbes[2]

The prevention of accidents from falls of roof and coal requires a considerable amount of personal judgment, for conditions are different even in adjoining working places in the same mine, and conditions may change from hour to hour even in the same working place as the coal is shot down and loaded. Experience has shown that many men can not or do not exercise enough care to protect themselves from injuries; moreover, where the foremen have been made directly responsible for the safety of the men under them, accidents from falls of roof and coal have, in many instances, been reduced as much as 50 per cent. There is now no question that careful, constant, and intensive supervision is essential in preventing accidents and more especially accidents of this class.

Falls of roof and of coal account for approximately 50 per cent of all coal mine accidents and annually result in the death of approximately 1,000 men and cause lost-time injury to approximately 50,000 men. These and other accidents are recurring with little or no diminution, usually not because of the lack of knowledge of safe and dangerous practices, or of the technique of safety, but largely because of the lukewarm interest of major executives in safety or the failure of mine officials to formulate and to enforce common-sense rules and regulations. When failure is the outcome of a sincere attempt of the managing officials to teach safety to the miner, the fault may usually be attributed to misguided effort.

Various solutions for the problem of eliminating accidents from falls of roof or coal have been suggested, some of which are: Standard system of timbering for the specified mining districts; departure from the room-and-pillar system to a more concentrated system of mining; more intensive supervision - that is, 1 section foreman for every 25 loaders instead of 1 section foreman for 75, 80, or even more loaders - the practice in most mining districts.

However, investigations conducted by the writer at numerous mines operated by a number of mining companies in several districts show that employees and officials alike are inclined to work under loose and unsupported roof, relying solely on the "drummy" sound and on their judgment and experience as to whether the roof will stay in place for one hour or for several hours. With but few exceptions, men who had been injured by falls of roof or coal or were found working under loose and unsupported roof, admitted that they knew of the possibly unsafe condition, but it was their opinion that the roof or coal was not loose enough to fall for some time.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6434."

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In many instances where miners were found working under loose roof without posts, the companies had rules requiring them to place posts in their places to within 6 or 8 feet of the working face. In some instances the rules also provided for an inspection by an assistant foreman at least once every two hours, proving that these coal companies intend that their mines shall be operated safely and efficiently.

The various investigations also revealed that as a rule, a plentiful supply of relatively good timber was available, and that few accidents from falls of roof or coal could be attributed to an inadequate supply of suitable timber. It was also found that officials and miners alike were generally familiar with the sizes of timber necessary and the various methods of placing them, so that lack of knowledge on this particular phase of mining can not be blamed for the high accident rate.

Many examples of delayed timbering and careless timbering were observed, such as timbering to within 20 to 30 feet of the working face when company rules provided that timbers should be within 6 or 8 feet of the face; no safety or center posts near the face when company rules provided that safety posts should be set before commencing work and should remain intact until the working place was cleaned up; no temporary or safety posts set prior to commencing work in connection with taking down loose roof on roadways or at the working faces; the use of chips of wood for cap pieces or the elimination of cap pieces altogether, and various slovenly practices in the setting of posts. These facts are known to practically every man engaged in the mining industry, and the writer is of the opinion that most accidents, both serious and fatal, resulting from falls of roof or coal, can be attributed to one or a combination of these causes.

It is obvious, then, that as long as men engaged in the mining industry, including mine officials as well as workers, continue to disregard such dangers and entertain their present attitude of apparent indifference or carelessness, so long will this particular hazard continue to exact its toll of human life. If the needless sacrifice of human life by falls of roof and coal is to be stopped or curbed, men must realize that a few good posts will hold up slate more surely than many long years of experience in guessing by sound or otherwise will determine how long loose overhead material will take to fall.

All loose roof must be considered as dangerous and as soon as it is detected it should either be securely timbered or taken down. The day of guesswork should have passed long ago; we should now attempt to profit by the sad mistakes of the thousands who have fallen victims to faulty judgment in testing roof or trying to determine how many hours the loose material would remain in place without timber support.

Many instances of men at work under loose and unsupported roof because they were not sure just where the track would be placed, or just how much room the trackman would need for the turn when he laid it, were observed, and on several occasions, officials were heard to give warning to "Watch that piece of slate! It is loose. Better set some posts after you load a few cars." But alas, in many instances, the advice went unheeded until too late.

Both miners and officials are at fault in following such practices. Under no consideration should men be permitted to work under loose and unsupported roof, loading out sufficient coal for the accommodation of a permanent post or while waiting for the track to be laid or the turn put in. The first duty should be the immediate protection of workers by the setting of temporary posts; the loading of coal to make room for track or permanent posts should be a secondary consideration.

In well-regulated mines timbering is done systematically and is made a necessary part of the day's work, and usually the result is that accidents from falls of roof are few and far between. Nothing is to be gained by failure to set posts in working places to within 6 or 8 feet of the coal face, and this is especially true where the places must be timbered ultimately. Obviously, the maintenance of posts close to the face works to the interest of the miner and company alike.

Men working at the face should know their jobs well enough and be sufficiently interested in their own personal safety to set posts without being continually told to do so. The supervisory officials should not only instruct them but insist that the first consideration be the setting of posts at regular intervals even when conditions are ideal, and the use of additional posts when conditions require more than the minimum provided by company standards. Some mining company officials claim that they are unable to induce their employees to set posts or follow out other safe practices; this is pure nonsense, and such statements are prima facie evidence of faulty supervision. It should be borne in mind that men engaged in mining are not different in any respect from men who work in mills, factories, or other industries, except that in the latter, well-defined operating rules and safety standards have been set up and all men are given to understand when they are employed and before they start to work that the rules and standards must be lived up to in every respect, and a sufficient number of competent supervisory officials are employed to see that such rules are enforced.

Without a doubt accidents from falls of roof or coal will be reduced to a minimum as will all other classes of accidents when mining companies decide to operate their mines as all well-regulated industries are operated. The adoption and enforcement of up-to-date rules and standards will not only aid materially to solve the accident problem, but will also increase the efficiency in every mining organization. It is obvious that the same rules and standards will not be applicable to all mines or even to more or less adjacent mines in any one county or district; each company should adopt its own rules and standards to suit its particular requirements. After rules and standards have been adopted, they should be rigidly enforced by a sufficient number of competent supervisory officials, and in general this requires that there be at least one "on the job" supervisor for approximately each 25 workers. Teach a man to be a careful and conscientious workman, and in most instances he will continue without great effort to take sufficient interest in his work to do it well and to feel justly proud of a well-kept working place. On the other hand, if the officials to whom he looks for guidance encourage, by silent consent or otherwise, carelessness and indifference with respect to the keeping of working places in an unsafe condition, the workman is extremely likely to be careless and indifferent and to lack the characteristics of a good workman. The officials, by proper conduct and good example, can do much toward cultivating safety habits among workmen; in fact, the supervisory officials are now acknowledged to hold in their hands the safety not only of the mine but of every individual who works in or around the mine; and there is no type of accident which occurs in coal mines which is so readily preventable through proper supervision as that due to falls of roof and coal.

SUPERVISION

What is effective supervision?

Supervision consists in seeing that the work in hand is carried out according to instructions or standard practice. This means checking men, materials, and equipment in such

a way that any errors of ommission or commission will be detected and corrected before serious damage is done. Supervision is only a part of a foreman's duty, but it is that part where contact is made with men, and it largely determines the success of the foreman.

With the expansion of industry, the foreman is generally recognized as holding the key position not only in efficiency but also in safety, since the maintenance of a competent, productive working force rests largely upon his ability to lead and inspire confidence in those under his direction. The success of safety work depends in large part upon his efforts to develop in his men a proper interest and attitude toward safe working conditions and methods. The attitude of supervisory officials toward safety is generally considered as the determining factor in the success or failure of accident prevention work. This attitude is directly influenced by the higher officials, through repeated indication by them of the importance they attach to accident prevention.

Experience has demonstrated that there is one fundamental principle underlying all effective accident prevention activities - namely, that safety is a definite and integral part of efficient supervision. A reduction of accidents, as a rule, can not be accomplished by the discharge of so-called careless men, as this usually necessitates the employment of new and untrained men who may be equally careless. This statement is attested to by the fact that some companies who keep accurate employment and accident records show conclusively that most of their accidents occur to men employed for periods of from one day to three months. A reduction in number of accidents is, to a very large extent, dependent upon the ability of the organization to obtain the proper cooperation from its existing force; this is efficient supervision, whether applied by the highest operating officials or the lowest-paid "straw boss."

If all of the employees were interested in and enthusiastic about their "jobs" and were willing to cooperate with one another, if all instructions were made perfectly clear to every one of the employees - then supervision could be reduced to the minimum. However, men's mentalities are such that what is clear to one may be confusing to another. Where one man puts his whole mind on his work, another may be thinking of outside pleasures or troubles. Where one man may think in terms of the whole organization, another may think only in terms of his own job or his private desires. Because of these various traits in man's character, someone must continually "check up" to see that instructions are understood and carried out.

The starting point of supervisory relations should be a sincere feeling on the part of the foreman that each man wants to do good work. That is a human characteristic which may not always appear on the surface, but it must be uncovered before any real progress can be made with any man. This means that tact and diplomacy must be a part of the foreman's equipment. Tact, to secure favorable reactions at the moment; diplomacy, in planning and doing the things that will develop the right relations in the future. Whenever a supervisor has contact with his men, he is in fact "selling" them his ideas or his personality; he wants the men to act in a certain way and his problem is the same as a salesman's problem when he is selling goods. Supervision is the part of a foreman's job that requires reliability - that is, every day faithfulness and dependability. A man with this quality may make a good supervisor even though it takes him a long time to acquire the knowledge necessary for the job. Such a man may not grasp new ideas as readily as some other man, but if he will stick to the details day in and out, he will be valuable.

Two men were found in a pillar working with only one post poorly set, where 14 posts should have been set to comply with company timbering standards. The section foreman admitted having visited these two men at least eight times during two days and on every visit had instructed them to set posts; he was of the opinion that he could not be held responsible since he had told the men to set the posts and the men agreed they had received such instructions but had not complied with same because they believed the roof was good and posts were not needed. However, when the section foreman was asked if he was paid to tell men what to do, or to really see that the various jobs were actually done, he admitted that he had been wrong and should see that his orders were obeyed. This is an illustration of lack of effective supervision.

Since attention and interest are so essential in securing good work, the supervisor can greatly lighten his task by paying attention to these requirements himself. An interested and satisfied man thinks constructively, whereas a disgruntled man will think destructively, even to the point of destroying himself or his opportunities. Therefore every contact with men should be tactful, and the best basis for this tact is the feeling that the man wants to do, or at least has it "in him" to want to do a good job if he is properly handled. On the other hand, if any of the negative qualities such as inattention, indifference, doubt, disgust, indecision, and inaction possess a man's mind, the result is very likely to be dissatisfaction. When dissatisfaction prevails, then to keep order, supervision may have to resort to discipline. In this day of independence, interest is a much better working tool than discipline, especially when the latter approaches (as it often does) tyranny.

The success of a foreman will depend in a large measure on his attitude toward the men under his supervision. If he is able to gain their respect and good will it will be far easier for him to keep the men interested and working to their full capacity.

A man's work may be checked and rechecked and the instructions repeated to him until it becomes almost second nature for him to perform his work in the proper manner, yet supervision must continue if periodical "slumps" are to be prevented.

Proper planning of work and the maintenance of a clean and orderly section or department - including, of course, clean haulageways and surface plant - are also essential to safety. A reputation for "square dealing" and a sincere interest in men also assists the foreman to maintain adequare discipline. Such qualities of character as honesty, sincerity, unselfishness, and charity enlarge and gain by being put into practice.

The same principles apply to our safety and accident prevention program. The more thought we give it, the more we must realize how unnecessary, distressing, and costly accidents are, and how easily they can be prevented with the exercise of reasonable care. The beginning always requires an added effort. To start is hard, but after a good start, the work of accident prevention becomes much easier. It is a matter of developing the mind along the right lines so that everyone can accept the full responsibility that goes with the "job." The development of the mind along the right lines depends upon the care given it and the amount and kind of thinking that is done.

The development of good supervision on the part of all mine officials is necessary if our coal mines are to reduce accidents, especially those resulting from falls of roof and coal. Supervision means not only giving instructions but also making certain that they are

fulfilled at the time specified, and generally that means at once. The following admirable set of rules, largely on supervision, are in effect in Alabama coal-mining regions, and the accident record of Alabama coal mines has been materially bettered in recent years due in large part to the enforcement of these rules:

REGULATIONS FOR PREVENTING ROOF-FALL ACCIDENTS

1. It shall be the duty of all company officials to exercise continuous, unremitting efforts to prevent the occurrence of accidents from falls of roof.

2. The strict observance and enforcement of all rules, regulations and laws for safety shall be a condition of employment for all underground officials and employees. And each employee, on employment shall be furnished a rule book and receipt taken for same.

3. Disregard of rules, regulations, and laws with respect to roof support shall be cause for discipline of any employee or official.

4. The official or foreman in charge of any section of a mine shall be held personally accountable for workmen in his charge who disregard the regulations as to care of roof and timbering and shall be subject to discipline.

5. A foreman or face boss shall not be placed in charge of a greater number of working places or men than he can visit with sufficient frequency during a shift to insure observance of the regulations. Such visits should not be less than twice per shift.

6. Strict adherence to a definite system of timbering, adopted by the management as suitable to roof conditions in the particular mine shall be compulsory. Additional timbers, necessitated by special conditions, shall be placed immediately as determined by the supervising official or by the worker.

7. Upon finding any portion of the roof in need of immediate attention, the supervising official shall remain and see that any dangerous material is either taken down or properly supported with timbering; or he shall order the workman to vacate the place at once and shall display a sign of danger at the approach to the place until such time as the roof is made safe.

8. The miners shall have suitable tools for setting timber. The company shall supply an adequate amount of suitable timber reasonably close to the point where the timber must be used.

9. Each accident due to a fall of roof shall be thoroughly investigated by a committee of underground officials, in which the official in charge of the district where the accident occurred, and all

other employees there at the time of the accident are to be included. The committee shall prepare a written report discussing the cause of the accident and include detailed sketches of the scene as well as placing the blame for the accident.

10. In the interest of safety, inexperienced miners should not be permitted to work alone; they must be placed so that they work with an experienced miner.

I. C. 6435 March, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

SAFETY CARS OF THE UNITED STATES BUREAU OF MINES

BY

J. J. Forbes and M. J. Ankeny

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I.C.6435.
March, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SAFETY CARS OF THE UNITED STATES BUREAU OF MINES[1]

By J. J. Forbes[2] and M. J. Ankeny[3]

One of the main functions of the U. S. Bureau of Mines, as authorized by Congress, is to investigate safety and health conditions, to disseminate information regarding safe and unsafe practices in mining, and to aid in improving safety and health conditions of mining communities and in promoting the welfare of the miner. Among several methods of reaching the miner and interesting him in safety work, perhaps the most effective is through the 11 mine safety stations and 11 railroad cars known as mine rescue, or more correctly, mine safety cars.

THE SAFETY CARS

Frequently, the Bureau of Mines receives requests for information regarding the purpose, equipment, personnel, and construction of its safety cars used in the field service. It is the intention of this circular to present to the mining public in particular, and to the public in general, some definite information regarding these safety cars and their functions.

The cars are operated throughout the mining fields of the United States and are headquartered as follows (see fig. 1): Car 1, Reno, Nev.; Car 2, Raton, N. M.; Car 3, Pittsburgh, Pa.; Car 4, Terre Haute, Ind.; Car 5, Pineville, Ky.; Car 6, Pittsburg, Kans.; Car 7, Huntington, W. Va.; Car 8, Duluth, Minn.; Car 9, Butte, Mont.; Car 10, Des Moines, Iowa; Car 11, Anchorage, Alaska.

Service at Mine Disasters

One of the most important purposes of the U. S. Bureau of Mines safety cars is to render service at mine fires and explosions where human life is endangered. State mine inspectors and other interested persons are supplied with itineraries of the car operating in their districts from time to time, so that they may know at all times where the car can be found in case of emergency. When these disasters occur the car may be reached direct or through its field headquarters, and upon receiving word of a mine disaster, or request for assistance, it immediately proceeds to the scene of trouble. Frequently a special locomotive is supplied by the railroads and in many instances the movement of the car is given the right of way over all other railway traffic. Upon reaching the destination, the facilities and personnel of the car are immediately available for rescue and recovery operations.

Following most mine disasters a thorough investigation is made by engineers of the Bureau of Mines, and a confidential report is submitted to the operator. The information gained in this manner from a large number of mine fires and explosions, together with the knowledge gained from the experimental work carried on at the Experimental mine of the bureau at Bruceton, Pa., serves as a basis upon which the bureau is able to recommend preventive measures whereby similar occurrences may be averted. The bureau car is often used as headquarters for the investigating engineer when such an examination is being made.

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2 Supervising engineer, safety division, U. S. Bureau of Mines.

3 Senior foreman miner, U. S. Bureau of Mines.

Safety Inspections

The Bureau of Mines safety car is at times used as a headquarters for bureau engineers when making safety investigations at mines. At the request of an operating company, or at least with its consent, bureau engineers make complete accident-prevention studies at mines, after which a report, also confidential, is submitted to the operator through the Director of the Bureau of Mines. In this report, explosion hazards and other dangers to life and property are pointed out, with recommendations as to how these hazards may be minimized. It is largely through studies of this kind that the bureau is able to keep in touch with the progress that is being made in the mineral industries in accident-prevention work, and the confidential reports made to the operators are used to disseminate the information which is assembled by the bureau's laboratory and field work.

First-Aid and Mine-Rescue Training

First-aid and mine-rescue training has been one of the main safety activities of the personnel of the Bureau of Mines ever since its inception. Until the past four or five years it was the practice to train selected groups of miners in first-aid so that they could render temporary relief and safely transport injured workers. In recent years, first-aid training has been gaining more and more recognition as being good accident-prevention training, hence, instead of training selected groups of men at an operation, the present-day practice includes the training of all or practically all persons at an operation.

To fulfill the current demand for first-aid training would be an impossibility if the old system of training only by bureau representatives were followed. During the past three years a cooperative plan has become popular, making it possible to fulfill in large part requests for training, and at the same time to train a much larger number of men each year. This cooperative plan, as it is commonly known, may be described as follows: A number of selected men from a mine or from several mines of a district are brought together in a class and given an intensive course of first-aid training by a bureau instructor, with the idea that from among this group of men those most adaptable will be selected to act as first-aid instructors. Considerable time and effort is expended in developing a number of instructors in this group. After this training has been completed the men organize classes in first-aid among the workers of their respective organizations and train them under the supervision of the Bureau of Mines representative. The classes are limited in size and generally are so arranged that each session of each class may be visited by the bureau instructor. The plan not only permits a much larger group of men to be trained at any individual mine or plant, but leaves a number of qualified instructors who are capable of maintaining the work after the car or bureau representative has left the district. The bureau cars are utilized to advantage for this type of work.

Many State-owned and privately owned mine-rescue stations are maintained throughout the United States. The personnel of the bureau safety car is often called upon to train the members of crews attached to these stations. In many instances, in which mines or plants are not equipped with oxygen breathing apparatus, progressive far-seeing managers desire to have men trained in the use of such apparatus for availability in case of emergency such as fire or explosion; the equipment on the car is available for this purpose and this phase of activity includes not only the large mines or plants but also the small mine or plant in the isolated localities. Mine-rescue training throughout the country has been fostered chiefly through the activities of the bureau's cars during the past 20 years and has now developed to such an extent that enough trained men are available in almost any mining region to take care of mining catastrophes which may occur in that region.

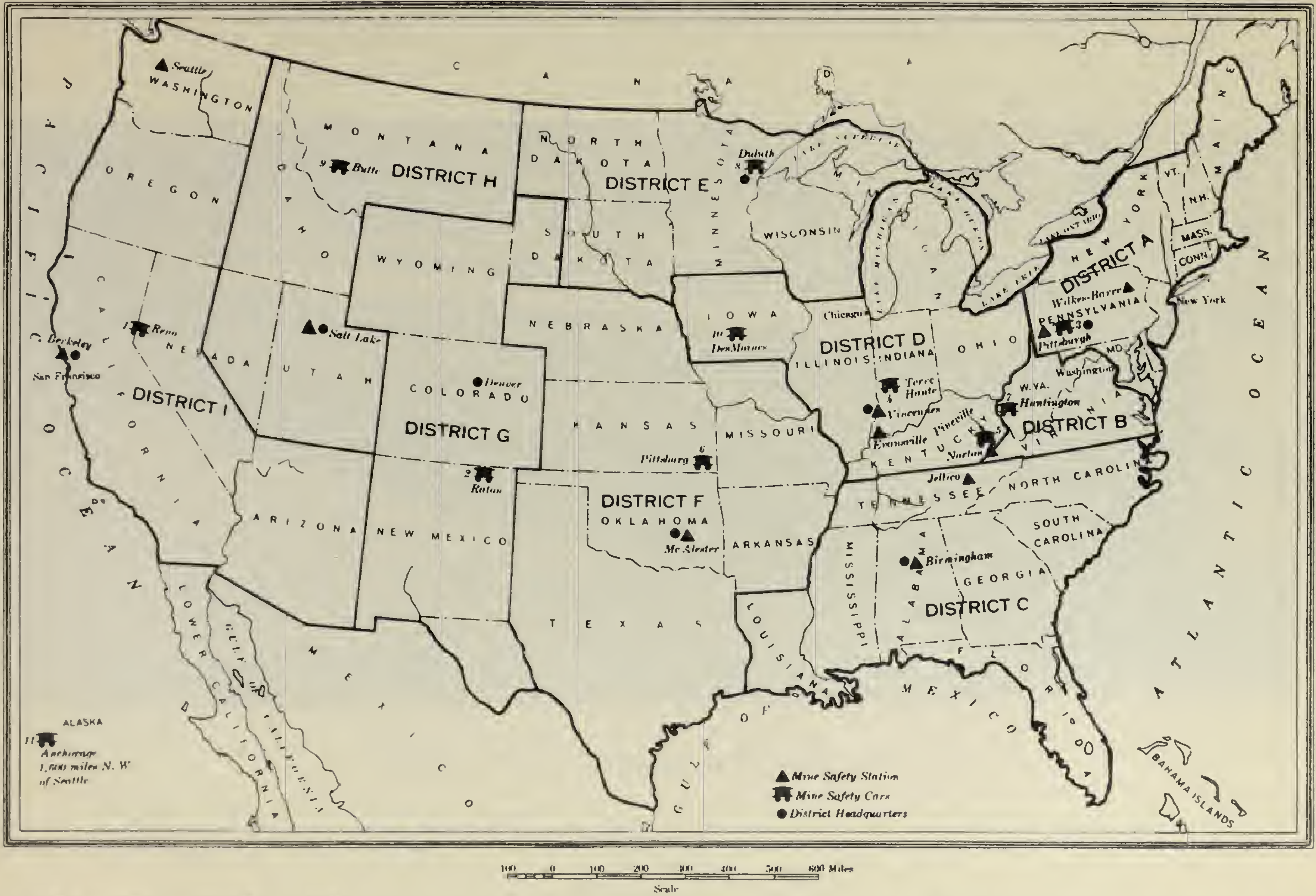


Figure 1.- Map of United States showing districts, district and mine safety car headquarters, and location of mine safety stations, safety division, U. S. Bureau of Mines

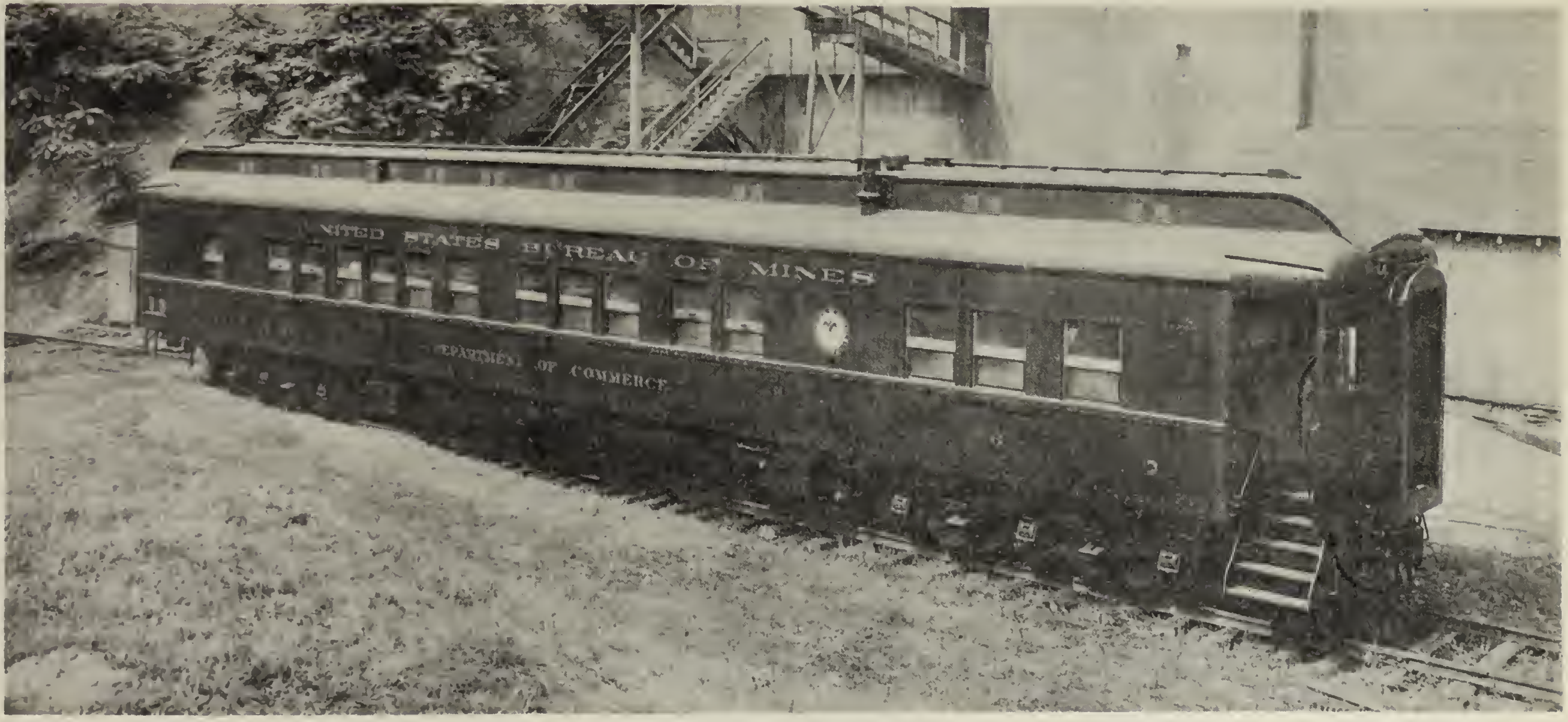


Figure 2.- View of all-steel mine rescue car

The Bureau of Mines has recently developed and put into the field a new course, known as "the advanced mine-rescue and recovery operations course, which was especially designed for mine officials and those who have had actual experience at mine fires and explosions. This course takes up in detail the use of protective and detective devices, methods of analyzing mine gases, and the technique of rescue and recovery operations following mine fires and explosions. The use of the bureau's cars aids much in the giving of this course.

In addition to the foregoing outline of work, the car personnel is often called upon to assist at local plant safety meetings, first-aid contests, and exhibits at State and industrial fairs; in fact, there are now fairs or other gatherings which annually use the Bureau of Mines cars as an important part of the exhibit.

Personnel and Equipment

The staff of the U. S. Bureau of Mines safety cars and stations is composed of carefully selected men who have had considerable mining experience or appropriate technical training; new employees are usually given a course of training at the Pittsburgh Experiment Station of the bureau and at the Experimental Mine at Bruceton, Pa., so that they are thoroughly familiar with the special features of the bureau's safety work and policies.

Car Engineer

Most of the Bureau of Mines safety cars have a qualified mining engineer in charge of the car; his duties are to supervise the work of the car personnel, to make safety investigations and reports of mining operations, to make disaster investigations and reports, to assist with local safety activities where the car happens to be stationed; and he assists at mine disasters and in many instances is called upon to take charge of the rescue and recovery operations following mine disasters. The car engineer is ever on the alert for developments concerning the safety or health of those employed in the mineral industries and is expected to transmit this information in the form of confidential reports to the Bureau of Mines at Pittsburgh and Washington. In many cases the information is given to the public in mimeographed or printed form.

Foreman Miner

The foreman miner is responsible for the upkeep of the mechanical and special equipment on the car and in the absence of the engineer he has full charge of the car and personnel. It is his duty to see that adequate materials and supplies are on hand at all times and he has supervision over the clerical work in connection with the operation of the car. He instructs classes in the use of self-contained oxygen breathing apparatus, in advanced mine-rescue and recovery operations, and in first-aid. He is also called upon to assist at mine fires and explosions and in some cases makes mine or other examinations and reports.

First-Aid Miner

The first-aid miner conducts and supervises classes in first-aid training and assists the foreman miner with his duties whenever possible.

Steward - Chef

Each car carries a steward-chef whose duty is to keep the car in a clean and sanitary condition at all times, make up the berths, wash windows, and prepare and serve all meals for the car staff.

Equipment

Complete mine-rescue and training equipment is carried on each car. The minimum amount of this special equipment includes 12 sets of self-contained oxygen breathing apparatus of a permissible type; electrically driven oxygen pump for charging apparatus; a number of cylinders of oxygen; 6 sets of all-service gas masks with an adequate supply of spare canisters; 2 oxygen inhalers to be used in connection with artificial respiration; 2 army stretchers; 4 canaries and 2 carbon-monoxide detectors; 1 Orsat apparatus for the analysis of mine atmosphere; 1 volumeter for the quick determination of combustible material in mixtures of mine dust; 6 permissible flame safety lamps; 12 permissible electric cap lamps; 1 life-line; 1 pyrotannic-acid carbon-monoxide indicator for the determination of small percentages of carbon monoxide in blood and air; 1 geophone; 1 sling psychrometer; 1 anemometer; and many other special devices. The car is also equipped with necessary household fixtures such as chairs, tables, linens, dishes, tableware, cooking utensils and cutlery, and at time of disaster eating accommodations frequently are provided for 20 or more persons at every meal.

Construction of Car

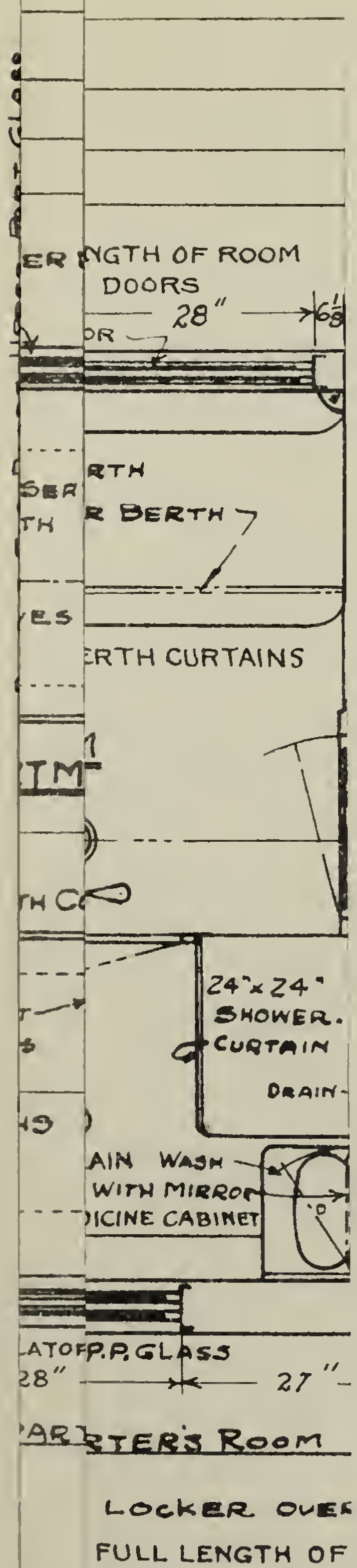
Formerly all safety cars of the U. S. Bureau of Mines were constructed of wood, but these old wooden cars have gradually been replaced by modern all-steel cars. At present all cars in the service are of steel construction, except the one operating in Alaska.

Car No. 6 is the newest car in the service of the Bureau of Mines. This car was completed and put into service about September 15, 1929. It was built by the St. Louis Car Co. in its shops at St. Louis, Mo. In the design and construction of this car are embodied many of the latest features of safe and effioient railroad car construction. Figure 3 is a plan of the car.

The car frame is constructed of heavy steel designed to meet the requirements of the Post Office Department for resistance to collision or overturning. The frame is practically proof against telescoping, as it consists of heavy steel castings and rolled steel shapes and plates. The Commonwealth cast-steel body bolsters and platforms which are used in connection with Commonwealth cast-steel ends for the body are the latest development of safety car building. The vestibule-end framing is of rolled steel structural forms and plates. The cast-steel platforms and bolsters are connected by deep fish-belly center sills and rolled "Z" and angle side sills. Vestibule end posts are set in pockets in the platform casting and are braced at the top by structural channels extending to the cast-steel end and to the side plate. This careful construction would make it necessary to destroy the vestibule frame before material damage could be done to the body end casting.

The sides are framed with "Z" and angle side sills, "Z" top plates, and structural channel side posts. The side sheeting is made up of copper-bearing steel as a preventive against corrosion. Posts and windows are placed to suit the various compartments of the car, which in turn are arranged to give a number of wide piers. These, in conjunction with the numerous cross partitions, make the car so strong that there is little possibility of damage to its contents except in the heaviest of collisions.

The regular car equipment includes a miner 5Px draft gear, miner class B-10 friction buffers, Westinghouse air brakes, Pullman pressure water system, Westinghouse automatic gas electric lighting, electric refrigerator, air compressor, and a dual hot-water heating system.



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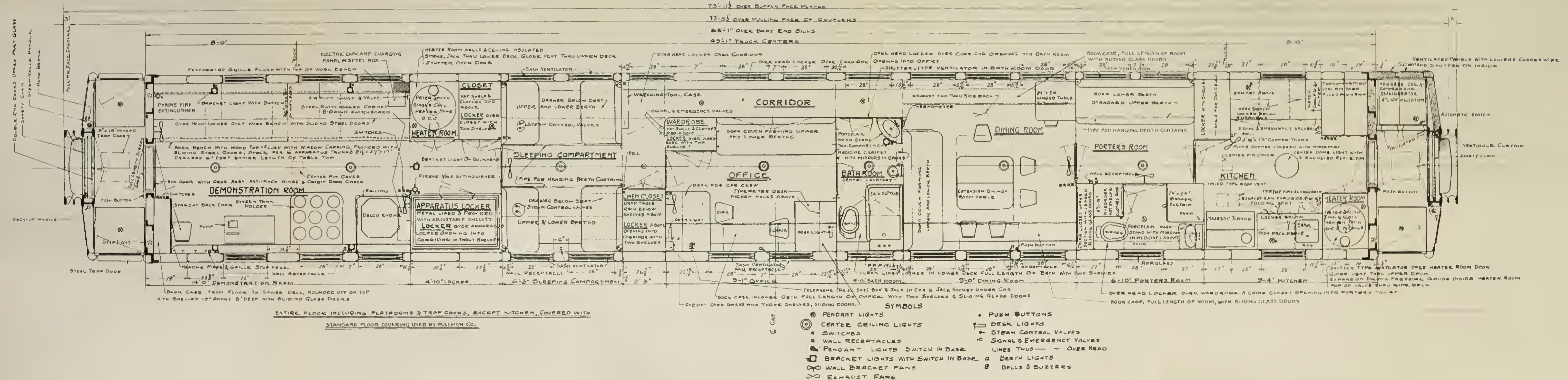

Figure 3.- Floor plan of all-steel mine rescue car

The interior of the car is painted in light, pleasing shades. The furniture is of mahogany. The interior is divided into compartments that experience has shown to be best suited to the needs of the service, as follows:

Demonstration Room

The demonstration room is equipped with metal tables, lockers, drawers, and cabinets to contain the apparatus used in demonstrating and instructing mine-rescue crews in the work carried on by the bureau. Complete equipment used by the staff in its mine-rescue work is carried in this part of the car at all times when it is not in actual use. In one corner of the demonstration room is a book-case which contains a complete set of bound volumes of publications of the U. S. Bureau of Mines. This library is invaluable to the car personnel for reference purposes. The demonstration room is also used as a workshop for the care and maintenance of the special equipment. The 100 cubic foot capacity oxygen shipping cylinders are kept in this room and in front of these cylinders, is the oxygen pump which is used to charge the oxygen breathing apparatus cylinders from the large cylinder.

Sleeping Quarters

The sleeping quarters for the staff adjoin the demonstration room. It is a double section, containing two lower and two upper berths. This section is arranged so that it can be used as a private conference room or lounge.

Office Compartment

The office compartment is located between the sleeping and dining compartments. It is equipped with an executive desk, a secretary's desk containing typewriter, bookcases, wardrobe, and a sofa couch which can be made into an upper and lower berth. A corridor is placed at one side of office so that the staff will not be disturbed by those passing from one end of the car to the other. On the dining room side of the office is the staff toilet, with an entrance from both the office and the corridor. The toilet room is equipped with a lavatory, small tub and shower bath, and flush hopper.

The dining room is entered from the corridor at the side of the office. It contains an extension table, chairs, folding table, china closet, bookcase, and a sofa couch which can be made into an upper and lower berth.

The porter-chef's room is located between the dining room and the kitchen. This compartment is equipped with a sofa couch, an upper berth, shower bath, lavatory, hopper, and wardrobe.

The kitchen is completely equipped for long runs and long lay-overs at mining plants. It contains a range, extra capacity coal bin, overhead water tanks, sink, dry food and vegetable lockers, work table, dish racks,lockers, and ventilating fans. The electric refrigerator is in the vestibule next to the kitchen.

The electrical system is so arranged that the car can be operated independent of any outside source of electricity; however, it is equipped with a transformer and suitable switching arrangements which make it possible to use 110 or 220 volt alternating current when available, making it unnecessary to operate the car power plant at such times. Special connectors with cable and plugs are provided for plugging in on local sources of supply when a car is in yards or at mine heads for extended stops.

The features outlined enable the car and crew to make long runs and stay at mining plants for long periods without undue fatigue or danger to health. The cars have traveled as much as 42,000 miles in a year, individual cars ranging from 1,500 to 8,000 miles.

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March, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SOME RUNAWAY CAR TRIPS ON INCLINES AT COAL MINES[1]

By J. J. Forbes[2] and M. W. von Bernewitz[3]

PURPOSE OF REPORT

In the United States there are hundreds of underground and surface (inside and outside) inclines at coal mines on which trips of cars are continually being run with coal, rock, supplies, and men. The safety of the men and prevention of damage to the slopes depend, excluding the personal element of the hoist-man, on the strength and condition of the wire cable, draw bars, and the couplings, and on the links of chains and pins as well. Within a recent period of four weeks there have been two runaway trips, one on the surface and one underground; it seems desirable to call attention of mechanical engineers and others at coal mines to the need for closer and more frequent inspection of the cable, draw bars, coupling links and pins, derailing devices, wheels, brakes, and other parts of cars or trips on which men are hauled on inclines. This bureau has never published anything on this class of accident heretofore.

How frequently runaway trips occur is not known. If no serious injury or fatalit has to be reported to the State authorities, a number of such happenings may occur and not be mentioned outside of the immediate vicinity of the mine.

Runaway trips can and do result in considerable loss of life, in injuries, and damage, either at or near the place of the break, or throughout a mine by causing an explosion of gas or coal-dust or both. This report is restricted to a study of the mechanical and physical phases of car trip construction and of the metal of couplings and ropes; derailed trips are not included and may be the subject of another paper. There is no need to mention gas accumulations, coal-dust and rock-dusting, open lights, protected power lines, and permissible equipment, because these have all been well covered in Bureau of Mines publications which have been widely distributed and partly or wholly reprinted by others.

ACKNOWLEDGMENTS

Most of the items regarding the runaway trips tabulated have been abstracted from reports made respectively by Clarence Hall, W.O. Snelling, and J. W. Paul; D. Harrington and J. J. Forbes; H. C. Howarth and G. S. McCaa; K. L. Marshall and E. H. Denny; J. J. Forbes, G. W. Grove, and R. D. Currie; S. P. Howell and K. L. Marshall; and J. W. Paul, H. D. Mason, and G. E. McElroy, all of the mining and safety divisions; and C. H. Herty, jr., and G. R. Fitterer of the metallurgical section of the Bureau of Mines. W. J. Fene, K. L. Marshall,

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and S. P. Howell of the safety division, and J. F. Lessig, mechanical engineer, and G. St. J. Perrott, superintendent, Pittsburgh Experiment Station, read the text and offered suggestions. W. J. McGregor, coroner for Allegheny County, and P. F. Nairn, State Mine Inspector of Pennsylvania, contributed valuable information.

DESCRIPTION OF SOME RUNAWAY TRIPS

Following is a chronological list of a few outstanding occurrences of runaway trips in various fields of the United States.

Typical runaway trips at coal mines

Year	Mine	State	Fatalities and injuries	Occurrence
			Underground runaways	
1907	A	West Virginia	361 killed (all in mine)	Coincident blow-out shot with wreck of 15 loaded runaway cars at foot of slope and consequent dust cloud which was ignited by open lights and caused an explosion.
1922	B	Alabama	90 killed (70 injured)	Runaway trip of three empty cars in rock slope produced a dust cloud which was ignited by an electric arc when the wreckage struck a power cable and caused an explosion.
1927	C	Pennsylvania	4 killed	Runaway trip of 36 loaded cars on semi-gravity plane raised a dust cloud which was ignited when wrecked car wheels came in contact with a power cable and caused an explosion.
1927	D	Colorado	7 killed	Runaway trip of 16 loaded cars was derailed by a drag; power cables and lighting wires were torn down, and the probable arcing (or possibly a carbide light) ignited the coal-dust cloud and started an explosion.
1929	E	Pennsylvania	46 killed (4 injured)	Runaway inclined apron-conveyor loaded with 60 tons of coal raised a dust cloud at bottom of slope and was probably ignited by electric arc, causing an explosion.
1930	F	Pennsylvania	3 killed (47 injured)	Runaway trip of 13 empty cars with men aboard ran 1,200 feet down a slope until wrecked.

Typical runaway trips at coal mines - Continued

Year	Mine	State	Fatalities and injuries	Occurrence
			Surface runaways	
1915	G	Pennsylvania	9 killed (17 injured)	Runaway trip of 18 loaded cars ran down 1,000-foot incline, left the tipple, and plunged into a group of 27 railway laborers.
1930	H	Pennsylvania	1 killed (1 injured)	Runaway trip of 15 loaded cars ran down a 40° incline, jumped the bin, and landed on loading space near main highway.

Details of Runaway Trips Listed

The following details may be added regarding the disasters listed:

Mine A. - At Mine A the trip had been hauled up the slope about 500 feet in length to a tipple at the opposite side of a river. The trip was partly over the knuckle of the tipple, and presumably, when the remainder of the cars was being pulled up, a coupling pin gave way. The free cars ran down the slope to the bottom and stirred up the dust. There was a derail outside of the slope, but an attendant did not act in time to prevent the cars from going into the mine. It was reported that there had been other runaways on this slope, but without injury to miners.

Mine B. - The haulage system in Mine B was as follows: Coal was transported in tight-gate wooden cars by electric locomotives to the "yard" or parting, which was in line with the slope and had scales, weigh-house, and two empty and two loaded tracks 500 feet long. The slope, about 6 feet high and 20 feet wide, had a pitch of 30° through rock, and intersected the coal bed at 850 feet. From the "yard" the coal was hauled in 5-car trips up the slope to the tipple and there emptied in a double 5-car rotary dump. Cars held 3,600 pounds of coal and weighed 2,200 pounds. Prior to the explosion, one of the cars of a 3-car trip became jammed in the guide rails of the rotary dump on the tipple. A chain block was unsuccessful in pulling back the trip. Then the two front cars were disconnected and pulled forward for a short distance with the hoisting rope disconnected; but in doing this, these two cars ran forward with considerable momentum, dislodged the jammed car, and broke a 60-pound rail which had been placed across the riding ring of the dump to prevent a possible runaway. Then the three cars started back into the slope (a main intake air course) and were wrecked near the foot after a run of more than 850 feet. The wreck caused the rupture of an electric power cable, the arcing of which ignited the coal-dust stirred up by the runaway trip; the resultant coal-dust explosion killed 90 and injured 70 others.

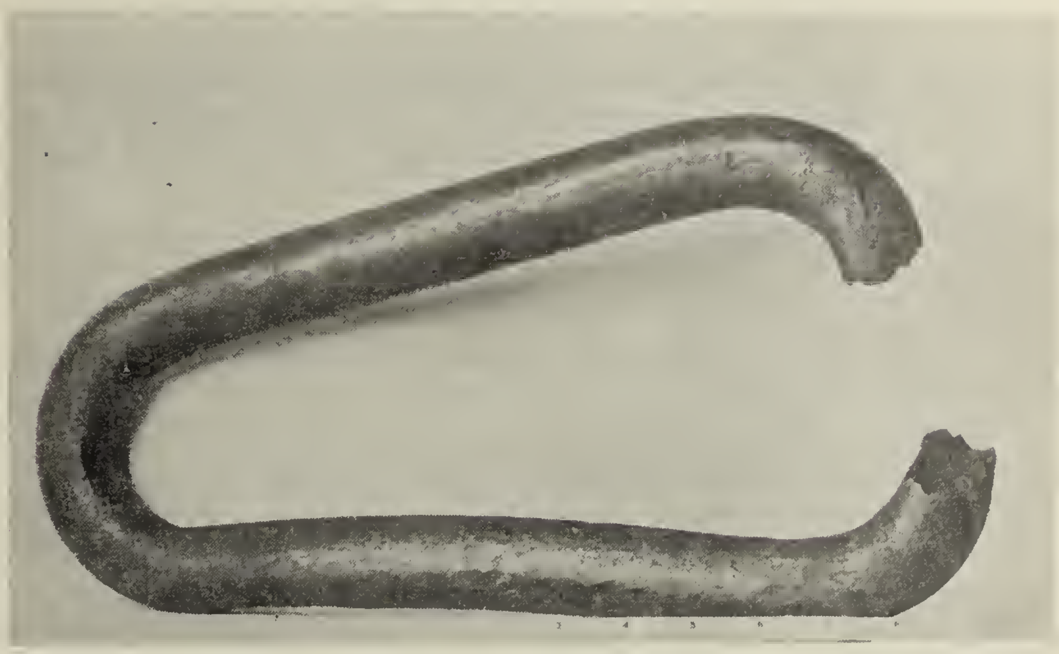

Figure 1.- Broken coupling link that caused the runaway in Mine C

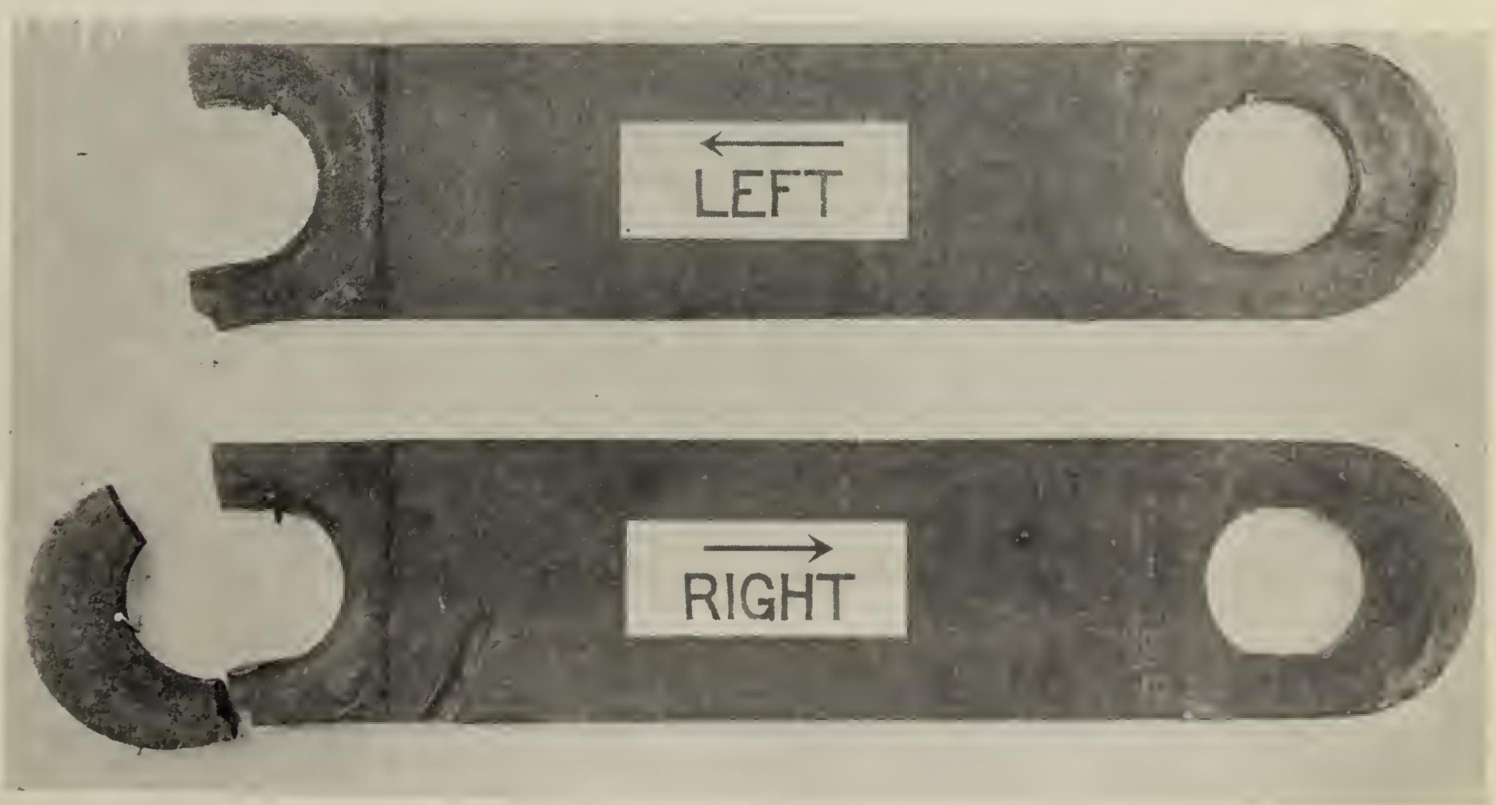

Figure 2.- Broken conveyor links from Mine E

Mine C. - The explosion in Mine C originated at the foot of a semigravity plane just off the main haulage-road about 6,000 feet from the mine mouth. At the time, a trip of 40 loaded cars was on its way down the north side of the plane; and a trip of 40 empty cars was going up the south side. This plane was doubled-tracked except for a distance of 350 feet at the lower end. When the loaded cars were at a point 1,200 to 1,500 feet from the starting place, the coupling link between the thirty-sixth and thirty-seventh cars parted and allowed 36 cars to run uncontrolled down the plane. Some of the cars were derailed at the end of the double track mentioned, and after traveling about 125 feet off the track they knocked out timbers, which started a fall of roof on the cars. The first 16 cars continued down the plane to the bottom; the other 20 cars were jammed in an entry above the fall and completely blocked it.

Figure 1 shows the broken link from Mine C, reported on by the metallurgical section of the Bureau of Mines. The link was of wrought iron and broke at the welded section; the angle of fracture was similar to that of the angle of weld. Microexamination (30 and 100 magnifications) revealed that the material of this link was extremely dirty even for wrought iron, which contained much slag. Chemical analysis gave 0.091 per cent phosphorus, 0.04 per cent manganese, 0.021 per cent sulphur, and 0.11 per cent silicon. The last three constituents were somewhat lower than in ordinary wrought iron, but the phosphorus content was nearly double that considered good; most of it was in the slag because the iron was highly oxidized. Unoxidized phosphorus is considered to be undesirable to have in any iron because it renders the iron brittle at normal temperatures. It may be concluded that this sample contained a great deal more highly oxidized slag than a good wrought iron; so much slag was present that failure occurred.

Mine D. - Mine D was opened by slopes from which entries were turned at 1,000-foot intervals on the strike of the coal bed. Immediately before the explosion, a broken hitching on the fourth car of a 20-car trip on the main slope allowed 16 cars to run back down the slope, a distance of about 125 feet. The trip had an iron drag on it, 2 by 2 inches by 4 feet in size. This lifted the first car from the track and threw it into a timber on the left side of the slope, after which the runaway cars stopped against the coal rib of a manhole. Power and lighting wires were torn down, and during the clean-up of the slope there was found evidence of arcing on the metal body of one car and on car irons of two cars with wooden bodies; one car hitching was almost burned through. This runaway produced a dust cloud which was ignited either by the arcing or, less probably, by an open carbide lamp worn by the trip rider. Seven persons were killed, and it is probable that had it not been for some rock-dusting done on the slope, the death list would have been greater.

Mine E. - Although the occurrence at Mine E is not strictly a case of runaway cars, it is one of a runaway conveyor carrying coal up a 30° incline 430 feet in length. This conveyor is of the apron type, is 5 feet wide, and when fully loaded holds about 60 tons of coal. It picks up coal from an 8-ton bin under the dump at the foot of the slope. At the time of the disaster the conveyor was full and broke near the head sprocket, rapidly running back the incline to the bottom of the slope.

Figure 2 shows two of the broken links from the apron conveyor at Mine E. These links were examined by the metallurgical section of the Bureau of Mines, but it is not definitely known that the initial fracture occurred in either of these two links. The coroner's inquest revealed that a few days before the disaster, the company's blacksmith repaired the conveyor by welding two links together to make an oversize link. In discussion

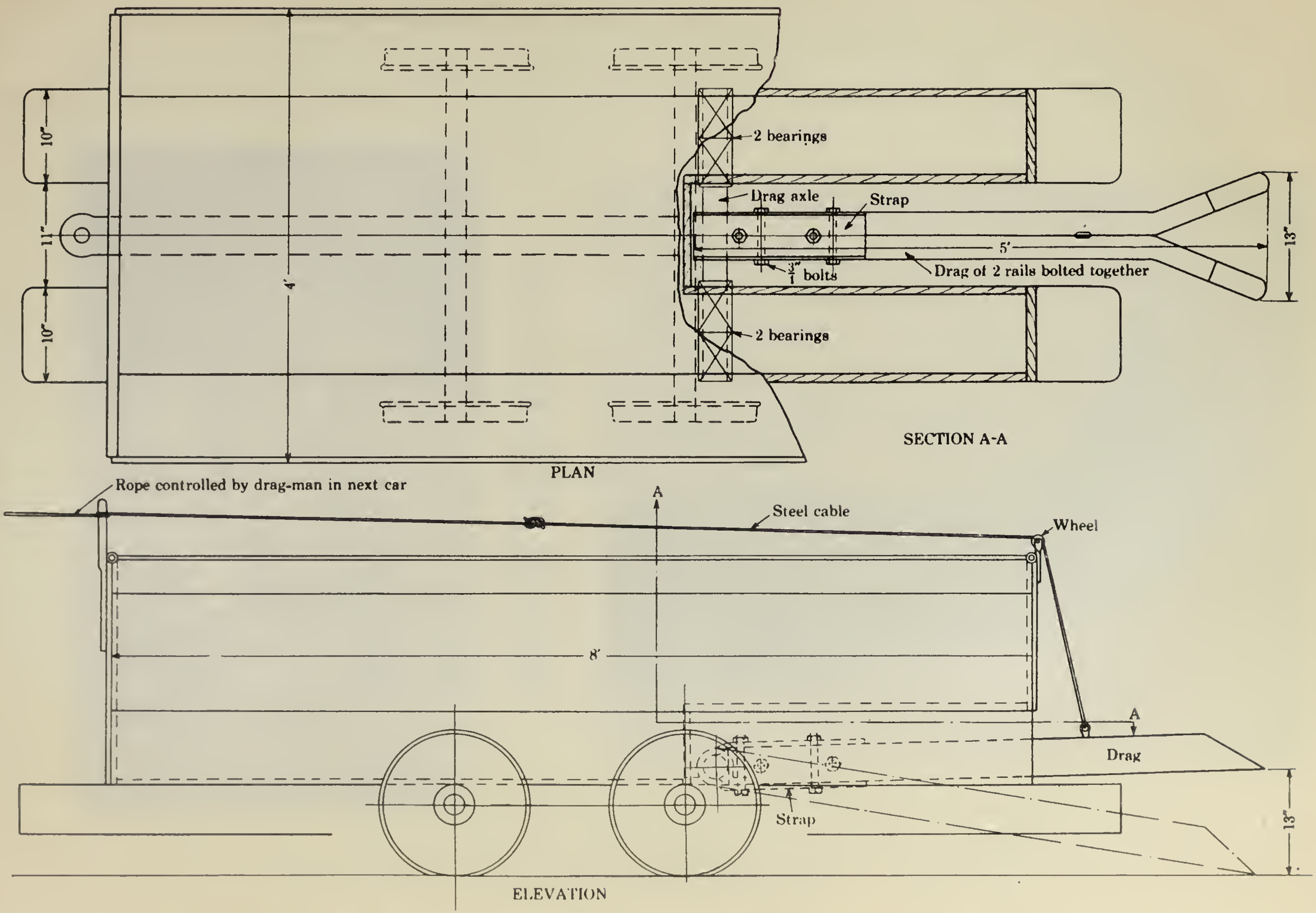

Figure 3.- Plan and elevation of drag car used at Mine F at time of runaway trip

Figure 4.- Part of coupling link that broke in Mine F. The slag streaks and high-carbon iron are clearly shown in weld and shank

Figure 5.- A safe coupling with split pin in clevis bolt

of this it may be said that although a weld may be strong in itself, the metal on either side of a weld may be materially weakened by the high temperature employed; hence, this repaired link might have been the initial cause of the runaway conveyor, some of whose links had been in service 12 years.

With regard to the links examined, appreciable amounts of impurities were found in the metal (for a steel containing 0.47 per cent carbon); these appear much like certain silicates resulting from deoxidation of steel with silicon.

Recrystallization had taken place in these chain links, as proved by photomicrographs (500 magnifications) of them and of a link that had never been used. Whereas the new link contained no extraordinarily large crystals of iron, the broken links showed large crystals, probably due to strain and aging. By heating links to 1,600° F. for 30 miutes and cooling in air, a treatment known as normalizing, a small crystal and more uniform metal, considerably strengthened, is produced; therefore, all links or chains which are subject to strain should be periodically normalized.

Mine F. - Fortunately, in this runaway there was not any explosion or fire, but 3 were killed, 47 others injured, and many men had to be extricated from the wrecked trip. The sequence of events is as follows: The day shift was going underground. The man trip had been placed near the top of the slope underground and consisted of 24 empty cars in front of which was a "drag-car." This slope was about 9,000 feet long and had a pitch of nearly 4°. The 190 miners were distributed so as not to exceed 10 men to a car, and a few were in the first two or three cars, but not in the drag-car. The trip had started down a short distance when the center link of a 3-link coupling between the thirteenth and fourteenth cars opened and permitted the front 13 cars and the drag-car to run down the slope. This part of the trip carried 95 men, about half of whom jumped out of the cars when they realized what was happening. The "drag-man" was in the second car, behind the drag-car, holding up the 100-pound drag by means of a cable and rope attached thereto. As soon as he noted the unusual acceleration he is reported to have released the rope which should have released the drag and should have stopped the trip. But the drag did not function in spite of further effort, and he jumped off the trip to outrun it. Eventually, after the trip had run down 1,200 feet of the slope the drag did work, tore up the track, and piled up the cars. Figure 3 shows the drag-car. This part of the slope had a pitch of nearly 5°.

A possible explanation of the failure of the drag to fall is that the rope which held the drag was not free or was fouled in some manner; also, it was disclosed that it had been the practice to hold up the drag with a fish-plate or other suitable piece placed over the bumpers when the car was moved in and out of its night storage place near the top of the slope. Although the evidence was to the effect that this had been removed on the morning of the disaster, it is possible that it had been overlooked.

Figure 4 shows part of the faulty link which was etched with hot hydrochloric acid by the metallurgical section of the Bureau of Mines. The link shows little evidence of wear; the fracture occurred at the weld, and the two adjacent sides appeared to have been separated for some time. An analysis revealed that the metal contained 2.34 per cent slag, close to the 2.5 per cent in normal wrought iron, and this slag contained exactly 10 per cent of silica and 90 per cent ferrous oxide, the correct composition. Figure 4 shows several streaks (white) of high-carbon iron (or steel). Microscopic examination revealed that this contained 0.20 per cent carbon, which is higher than in normal wrought iron. As wrought iron

and steel do not weld readily, this defect may have caused a poor weld. However, the failure of the link was primarily due to a poor welding practice, aided by nonhomogeneous iron, and possibly by inadequate inspection.

Mine G. - At Mine G there was a new, single-track surface incline from the mine to a new tipple on the railroad, a distance of 1,000 feet with an average grade of 16 per cent. The tipple was at an angle to the right from the tipple approach or trestle. Not far above the trestle, the incline branched to form one track for loaded cars and one for empties, and just below this branch was a safety switch. A new $1\frac{1}{2}$-inch cable attached to a 6-foot drum driven by a 75-horsepower motor hauled the loaded and empty cars. On the afternoon of the accident, a trip of 18 loaded cars had just started down the incline, when the attachment broke between the last car and the haulage rope, and the entire trip of cars ran away down the incline to the tipple. The safety switch near the tipple approach was thrown over by the switch-tender, but it failed to derail the flying trip which swept the full length of the tipple approach and then plunged off the tipple just beyond the point of curvature in the track.

Four men were working on the tipple; two of them managed to escape with slight injuries, but the other two were caught and crushed to death near the dump, where the runaway trip sideswiped the structural steel frame. Forty feet below the tipple, 27 men were ballasting the railroad tracks at a point in line with the approaching trip, but at least 100 feet distant horizontally from the place where the trip leaped from the tipple. The rushing trip of at least 45 tons leaped this entire distance and caught many of the men as it fell. Seven were killed.

An investigation revealed that a break in the clevis to which the trip was coupled had apparently permitted the loaded trip to get away. A piece about 4 inches long had been broken from the eye of the clevis, thus allowing the coupling to slip out and free the trip. The clevis was made from $1\frac{1}{2}$-inch iron and was strongly socketed to the $1\frac{1}{2}$-inch cable. At the point where the fracture occurred there was an inner defect in the welding; the outside rim, however, appeared to be smooth and substantial, so that it would have been difficult to detect this defect by visual inspection.

Mine H. - The situation at Mine H was somewhat similar to that at Mine G, in that the incline slopes toward a well-traveled highway instead of a branch railway. The evidence given at the coroner's inquest and investigation by the State mine inspector revealed that late in the afternoon there was a trip made up of one car of slate and 14 cars of coal and rock. The cars held about $1\frac{3}{4}$ tons each. The first three cars were on the outside of the pit mouth, the first one being about 60 feet from the safety blocks of the incline plane, which had a 40° grade and was double-tracked. For 90 feet between the pit mouth and the safety blocks of the incline and for another 210 feet in the mine there was a down grade of 2 per cent. The cars were coupled and it was stated that the first seven cars had the brakes set and had 12 sprags in the wheels. The slate car was cut off and dumped on the slate pile 150 feet distant, but as this car was being returned, the mule-driver noticed the trip in motion, sliding along toward the incline plane. He spragged some of the cars and set the brakes on about four of the rear cars, but this did not check the trip, which was derailed at the safety blocks. The 14 cars did not pile up at that point because of the dry and hard condition of the ground, and continued running down the hill. At one point some of the cars were 15 feet from the tracks, but here they struck a bank and veered back onto the track be

and some of them continued to the tipple. The trip broke on the way down and only 7 cars reached the tipple, one end of which they ripped through; they demolished the weigh office, which was 60 feet in front of the tipple. The shipping clerk was in the weigh office at the time and was struck by flying material which fractured his skull, resulting in death on the following day. In the wreckage the mine superintendent found eight cars with brakes set and nine cars with sprags in; the other cars were so demolished that he was unable to determine whether the brakes were set or if they were spragged.

RUNAWAY CARS IN ROOMS AND ENTRIES

Many rooms in mines are driven to the rise and many on the dip of the coal. In either of these places, single cars have run away and caused injuries to men any where along the track between the room-neck and the face. In either condition a large block should be placed under the wheels, and sprags should be put through them. For a steeper pitch a metal clamp should be attached to the rail to prevent the block from slipping. In addition, a two-pronged iron drag is a safeguard for cars on steep grades. Between 30 and 40 men a year are killed in the United States by car or trip runaways.

RECOMMENDATIONS AND CONCLUSIONS

Engineers of the Bureau of Mines who investigated these and other accidents made the following recommendations as far as haulage was concerned:

1. The suggestion was made for Mine A that derail switches should remain open except when held closed by an attendant during the passage of a trip of cars entering a mine.

2. Derail devices should be placed on the approach to a tipple to prevent, as far as possible, runaway trips from rushing down a slope. This device could be operated by an attendant employed on the tipple. Another derail device should be placed in a slope approximately 100 feet from the point where it intersects a yard or parting, as in Mine B.

3. To prevent such a recurrence as in Mine C, suitable provision should be made for keeping trips of cars under control at all times, or if a coupling or rope breaks, derailment of cars should take place before they have gained much speed. It was suggested (a) that there be systematic inspection for defective couplings, draw bars, and ropes; the two former may be examined by hammer tapping; (b) in addition to strong couplings and pins, all cars run on inclines should have strong safety chains, kept in use at all times, whether on an inclined or level road; (c) derail devices placed at or near the top and at or near the foot of a plane, and controlled from the head-house by the plane-runner, may be used to derail cars in emergency.

4. In the case of Mine D it was proved that a trip derailment on a slope of moderate pitch is likely to raise a cloud of coal-dust, dense enough to be ignited by an arc and cause an explosion. The ordinary drag may not derail the trip until it has gone backward 100 feet or more, and the momentum gained in such distance may be sufficient to cause violent impact and produce a heavy dust-cloud where the cars are wrecked. It was recommended that car couplings receive both visual and mechanical tests at least at 3-month intervals to determine freedom from defects. The couplings should be at least the same diameter as the haulage rope. Safety chains should be attached to all cars and the chains should be used at all times on both loaded and empty trips, whether on a slope or on level track.

5. No means were provided at Mine E to prevent the conveyor from running back down the incline in case of a break, but there should be some device or devices for doing this.

6. As the drag was unworkable in Mine F at the critical moment, a change in the means of its support was recommended and made. This improvement takes the weight off the control rope, because the drag is now supported by a pin and lever attached to and easily operated by the rope. Other recommendations included the use of special man-trip cars to reduce the hazard of runaways; that brakes be maintained in operating condition at all times; and that a cable through and attached to each car and secured to the haulage rope well above the socket be installed.

7. Frequently, a coroner's jury finds a verdict of accidental death if men have been killed and the cause can not be definitely attributed to anything; seldom is any apparent interest taken or remarks made regarding the mechanical features of a runaway trip. Broken links of couplings are exhibited, but nothing is said concerning the weld. Derail switches are mentioned, but their workability is lightly considered. Broken wire ropes are the prime cause of some runaways, but their condition is hardly questioned. Of course, these features of a runaway are really for action by the State mine inspector, yet the coroner's jury should be alive to their importance.

8. The examination of the chain links described leads to these conclusions: (a) That none but the best wrought iron or steel rod or rounds should be purchased and made into links; (b) that the welding scarf should be properly shaped; (c) that the fire should be clean and a proper welding temperature reached; (d) that if coal is used in a forge it should be low in sulphur to prevent red-shortness in the iron; and (e) that a good flux such as borax be employed, not sand, as is sometimes used.

9. With regard to links for conveyors, none but the best steel bar should be used for repairs and renewals. Holes for the pins or bearing ends should preferably be drilled, not punched.

10. Every trip hauled up a slope should have a loose drag on the rear car to prevent its running back; conversely, every man-trip lowered down a slope should have a controlled drag on the front car.

11. All concerned in a haulage system should be sure that the draw bar, rope, hitchings, couplings, pins, safety chains, derail, and drag are in place, are secure, and in good physical condition.

12. There is a definite need for a reliable automatic derailing device wherever there is slope haulage. The adoption of an automatic derailer not only increases the safety of employees, but it also increases the economic operation of slope haulage by decreasing property damage. A derailing switch of this type, electrically operated, has been installed on the slope joining the mine and tipple a short distance above the tipple landing of Roslyn No. 3 and other mines of the Northwestern Improvement Co., Roslyn, Wash. Information Circular 6226 (January, 1930) of the U. S. Bureau of Mines, by S. H. Ash and R. H. Kudlich, describes this reliable derailer.

13. Hoistmen, whether at steam or electric hoists, should not jerk the haulage rope when it is attached to a trip. Such treatment may cause the breakage of the rope, and also of a chain or pin or other part of a coupling or possibly the draw bar.

14. As a single-link forged coupling is as safe mechanically as any car hitching, its use is recommended, although a single link is not as flexible as a chain for going around curves.

15. Chains of three or more cast alloy-steel links are procurable and are recommended.

16. It is suggested that as rails are examined by X-rays, the links of coupling chains could also be examined in this manner by makers of chains. Defects would thus be revealed.

TREATMENT AND CARE OF CHAIN

In its NATIONAL METALS HANDBOOK for 1930, the American Society for Steel Treating gives recommended practice for the heat treatment and care of sling and crane chain, and the following items are abstracted therefrom:

1. The reliability of chain depends upon the quality of the welded links and the metal from which they are made.

2. Wrought iron for chain usually has the following approximate chemical composition: carbon, less than 0.05 per cent, manganese, less than 0.10 per cent.

3. The average physical properties of wrought iron for chain are as follows: Tensile strength, 46,000 pounds/square inch; yield point, 23,000 pounds/square inch; elongation in 8 inches, 26 per cent; reduction in area, 40 per cent.

4. To judge the workmanship of a welded fagot, a cold feathering test may be applied by flaring the end of the bar under a hammer. This should show a high degree of ductility and a fibrous, silky fracture free from bright spots.

5. Wrought iron and mild steel should not be mixed or welded together, because they require different temperatures.

6. If a weld has been properly done, the welded portion may not need a normalizing treatment to refine the structure, but with large chain, portions of the link adjacent to the welds have been treated to or near the welding temperature, and if not hot-worked, a weak, coarse, crystalline structure is developed that requires a normalizing treatment for refinement. To do this, the chain is placed in an annealing furnace and covered with a sealed sheet-iron hood to prevent scale formation, until the complete charge is at a temperature of 1,200 to 1,400° F., at which the chain is kept 45 minutes. The time for heating is about three hours. The chain is then withdrawn and allowed to cool in the air.

7. Overloading is probably the most serious service abuse to which chain is subjected. A new chain may be impaired by overloading the first few times it is used. If a chain is carefully measured at frequent intervals, any elongation that is not the result of wear is sure proof of overload.

8. Improper application of a chain such as a sudden load, or jolt, twisting, kinks, and bending, sets up stresses that are far in excess of those caused by the weight of the load lifted.

9. A safe load for a chain of 1-inch diameter metal is nearly 17,000 pounds; for $1\frac{1}{8}$ inches, 20,000 pounds.

10. Don't forget constant and careful inspection.

DETERIORATION AND INSPECTION OF WIRE ROPES

Wire ropes or cables for car trips are subjected to more or less abuse by dragging, jerking, stretching, kinking, weather, and corrosive agencies. Some of this treatment is avoidable, but under normal circumstances a wire rope starts to deteriorate as soon as it is placed on a hoist reel and attached to mine cars. Some ropes drag on the bottom of a slope or incline; others ride on idlers and rollers. When not in use, ropes should not be allowed to lie on the ground; they should be either coiled on the hoist reel or hung up, as is sometimes done to prevent contact with acid water. Wire ropes should be inspected frequently--daily, in fact--and lubricated at frequent intervals. Occasionally the interior should be examined by untwisting, and if the core is dry, the rope should be cleaned and oiled. The socket or eye and clips by which a rope is attached to cars should also be examined for broken wires and other faults. If any are found, the fastening should be cut off and another one attached. If a rope shows an excessive number of broken wires or wear in its length it should either be discarded or a piece cut out for a test of its strength. Where there is any doubt, even by visual inspection, it is wise to discard a wire rope.

COUPLING PINS

Defective coupling pins are a source of accidents in mines. Frequently, badly bent pins are kept in use. Sometimes, they are not inserted properly and result in runaway trips. Coupling pins should be inspected regularly, changed if necessary, and straightened if not bent too much. The pin should be provided with an automatic device to prevent its slipping out of place.

SPRAGS

Wooden sprags are in more or less common use in car wheels underground and on the surface; a short piece of light rail is also suitable. A plentiful supply should be available at all times. If they are too short, men are liable to have their hands injured; if they are too long, sprags project from a car and are liable to strike anyone who might be standing close by. Spragging of cars in lieu of proper blocking is not good practice unless the pitch is low.

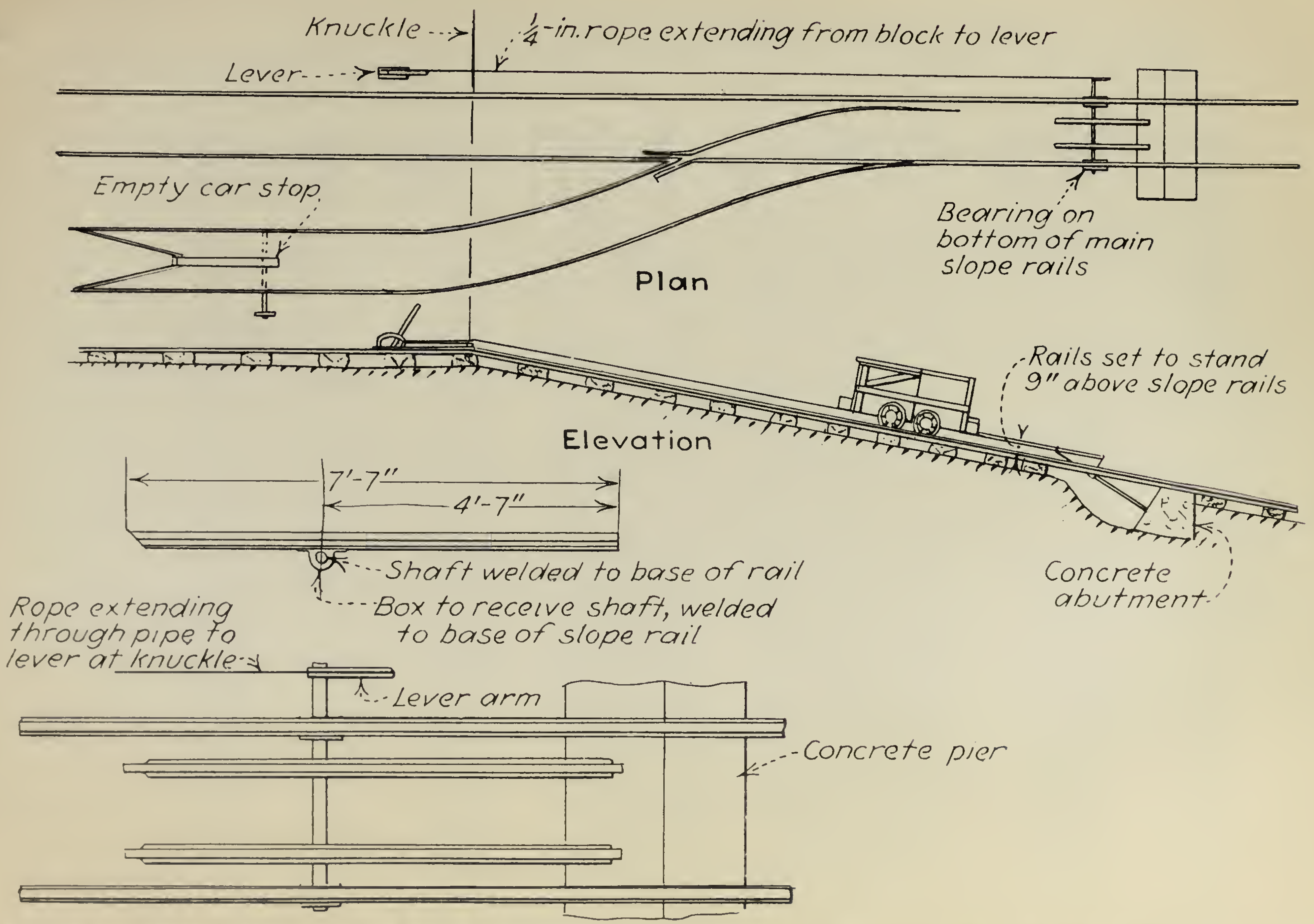

Figure 6.- Safety block at slope entrance to stop possible runaway cars. It must be held open while trips are being lowered into a mine

SUGGESTIONS FROM THE BUREAU OF MINES ACCIDENT-

PREVENTION COURSE (1930)

Section 4, Haulage Accidents, in the Accident-Prevention Course of the Safety Division of the U. S. Bureau of Mines, now being used in the field, contains the following items:

1. Car brakes, when not properly maintained, constitute serious hazards, such as failure to control cars by depending on the faulty brakes to hold, and failure of some part of the brake rigging while attempting to set brakes may cause the trip-rider to lose his balance or be thrown under the moving trip.

2. There is a large variety of coupling devices in use. None affords complete safety if the coupling is done while cars are in motion, itself a dangerous practice. The principal designs are link and hook; link and pin; two hooks and either single link or multiple link; clevis and links or rigid bar, and male and female design. (See fig. 5.) There are two inherent dangers in all coupling devices--breakage and uncoupling. Breakage can be minimized by selecting suitable sizes, shapes, and metal; discarding the couplings when worn, and inspecting with heat treatment at intervals.

3. To avoid accidents from breakage of the main coupling, an additional sturdy coupling of the car to the rope above the main coupling is frequently used. To avoid accidents caused by one or more cars breaking loose, a 1-inch wire rope is sometimes run above or under and fastened to each car of a man trip. In some western coal mines, all pit cars have safety chains in addition to and independent of the draw bars and couplings.

4. Drags are much used on loaded trips on slopes, and on grades in locomotive haulage. To prevent the drags from knocking out rollers, the drag may be placed to the side rather than to the center of the car.

5. Every producing entry which is on a grade should have a runaway switch, outby the first working or producing place, to prevent runaway cars from running back down the grade.

6. A safety block such as shown in Figure 6 should be provided at slope entrances to stop possible runaway cars and trips. Of course it must be held open when trips are being lowered into a mine.

I. C. 6438

May, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

INDEX TO
GEOPHYSICAL ABSTRACTS
NO. 1 TO NO. XX

COMPILED BY

PALMER LARSEN

I.C. 6438
May, 1931

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

INDEX TO GEOPHYSICAL ABSTRACTS[1]

NO. 1 TO NO. 20

Compiled by Palmer Larsen[2]

1. Gravitational Methods

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6438."

2 - Mining division, U. S. Bureau of Mines.

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4. Electrical Methods

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EFFECT ON WORKERS OF AIR CONDITIONS

BY

R. R. SAYERS

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

EFFECT ON WORKERS OF AIR CONDITIONS[1]

By R. R. Sayers[2]

That the maintenance of proper air conditions is a most important factor in the control of occupational hazards is shown by the large amount of material presented each year through the technical journals on the effects on workers of injurious dusts, gases, and variations in temperature and humidity.

SUMMARY OF RECENT LITERATURE ON EFFECTS ON WORKERS OF EXPOSURE TO DUSTS

The subject of the control of dust and dust diseases continues to be a matter of serious concern in most of the industrial countries of the world, as indicated by the following summary of reports published in a number of foreign countries as well as in the United States.

Australia.- The Workmen's Compensation Act of 1920 for New South Wales[3] provided for the promulgation of schemes for the compensation of workers disabled by silica or other dust. In 1924 the prevalence of silicosis and tuberculosis among Sydney stonemasons, quarrymen, sewer miners and rockchoppers was investigated by a Technical Committee of Inquiry, appointed upon the recommendation of the New South Wales Board of Trade. The findings of this committee were published by the Government in 1925. No scheme was promulgated by the Government and in 1926 the Act was amended limiting its scope to workmen disabled by disease caused by silica-dust. This was promulgated and gazetted on September 16, 1927, and applies to men working in sandstone as stonemasons, quarrymen, rockchoppers, and sewer miners in the County of Cumberland.

1 Report of the Chairman of the Subcommittee on Physiological Factors of the Committee on Ventilation of the American Institute of Mining and Metallurgical Engineers presented at the meeting in New York City, February, 1931. The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6439."

2 Chief surgeon, U. S. Bureau of Mines or, chief, health and safety branch; surgeon, U. S. Public Health Service.

3 Third Annual Report and Statistics of the Workers' Compensation Commission of New South Wales for the Year July 1, 1928, to June 30, 1929; Published December 18, 1929, 78 pp.

The report of the Workers' Compensation Commission[4] gives information in regard to 1,300 of 1,500 workmen examined up to August 23, 1929. Of 107 stonemasons examined, 9 had tuberculosis, 17 were in various stages of silicosis, and 26 had silicosis with tuberculosis. These men had worked in the industry for periods varying in length from 10 to 55 years. Of the 29 monumental masons, 2 had tuberculosis, 1 had Class A silicosis, and 5 had silicosis with tuberculosis. No statement is given as to the length of time worked by these men, with the exception that the one suffering from Class A silicosis had worked for 31 years in the industry. Of the 37 ballast quarrymen, 1 had tuberculosis and 2 had marked silicosis. The two with marked silicosis had worked 2 and 12 years respectively, but no statement is made in regard to the one man with tuberculosis. Of the 80 dimension quarrymen, 6 were diagnosed as having tuberculosis, 3 as having early silicosis, 2 as having marked silicosis, and 16 as having silicosis with tuberculosis. The three cases diagnosed as in the early stage of silicosis had worked 18, 20, and 20 years, respectively, and those with marked silicosis had worked 16 and 20 years, respectively. Among the 377 rockchoppers examined, 12 had tuberculosis, 25 had silicosis, and 10 had silicosis with tuberculosis. These men had worked in the industry for periods varying from 5 to 32 years.

Workmen in the Broken Hill mines are compensated under a scheme contained in the Workmen's Compensation (Broken Hill) Act, 1920-1927. The Broken Hill (Pneumoconiosis-Tuberculosis) Compensation Scheme is administered by a joint committee at Broken Hill.[5]

In regard to conditions at Broken Hill, according to George,[6] up to June 30, 1929, in no case has a man with no mining experience before 1921, and who has worked in Broken Hill only during the succeeding years, been discovered with dust disease of the lungs. This means that at Broken Hill, as far as is known, the period of exposure is at least eight years, and the appearance of the first case of pneumoconiosis among men in this classification is still awaited.

The idea that two factors operate in the development of pneumoconiosis in a miner is considered by George as probably the most important development during the past few years toward an understanding of the true nature of dust disease of the lungs. He states that the following observations made at Broken Hill during the past eight years support the view that there is a tubercular element in every case of dust disease, even in so-called "simple" pneumoconiosis:

> The fact that of many men exposed at work to exactly similar conditions only a percentage are affected by dust, i.e., there must be an additional factor in the man himself for the production of the disease.

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4 See footnote 3.
5 See footnote 3.
6 George, W. E., Eight Years' Further Observation of Dust Disease of the Lungs Occurring Amongst Miners at Broken Hill: Third Annual Report and Statistics of the Workers' Compensation Commission of New South Wales for the Year July 1, 1928, to June 30, 1929, Dec. 18, 1929, 78 pp.

The large percentage of men with "simple" pneumoconiosis who have eventually developed "overt" tuberculosis, even when removed to surroundings where it was most unlikely they would be reinfected by the tubercle bacillus. This is particularly emphasized by the after histories of certain Broken Hill miners (34 in number), suffering from simple pneumoconiosis, who took up blocks on the Murrumbidgee Irrigation Area at Griffith, New South Wales. These men cleared virgin country, erected new homes, and lived in the most healthy surroundings possible, but 12 of them have since developed overt tuberculosis, and 7 of the 12 have died of pulmonary tuberculosis. These men, far from Broken Hill, were not in contact with beneficiaries who had tuberculosis, and who may have been accused of bestowing the tubercular infection.

In most radiographs, even of so-called "simple" pneumoconiosis, an area suspicious of either healed or incipient tuberculosis can be recognized if searched for. Gradually improved radiographic technique has made the recognition of this area much more frequent than was the case eight years ago. When watched over a period of years, this suspicious area has frequently been observed extending, and eventually the condition of pneumoconiosis with tuberculosis is established clinically as well as radiographically. In cases which have suddenly developed an acute tuberculosis after showing no change for years an area such as this has been observed to be the starting point of the active disease.

Chapman[7] has the following to say in this connection:

> In the production of silicosis it would thus appear that the particles of silica are in many individuals absorbed in the lungs without giving rise to silicosis. In other individuals the particles of silica do not appear to be absorbed, but give rise to inflammatory areas starting silicosis of the lungs. It has been suggested that this inflammatory process occurs in persons already the subjects of tubercular inflammation. It is said that it is the tuberculous subjects who become silicotic. The question may be asked whether the areas of the lungs in which particles of silica are not removed and in which silicosis develops are not already the seats of tuberculous processes. The answer would, however, appear to be in the negative since the characteristic feature of the silicotic inflammatory change is its uniform distribution throughout the lungs; even at an early stage, uniformity of distribution in the pathological process is characteristic of silicosis. On the other hand, it is unusual to observe uniformity of distribution throughout the lungs in the early tubercular inflammations. My friend, Dr. S. A. Smith, reminds me that early tuberculosis tends to attack the apices of the lungs. Many have observed that the apices tend to be the last areas attacked by silicotic inflammation.

7 Chapman, H. G., Silicosis and Tuberculosis. Third Annual Report and Statistics of the Workers' Compensation Commission of New South Wales for the Year July 1, 1928, to June 30, 1929, Dec. 18, 1929, 78 pp.

England.- Bridge,[8] in calling attention to the importance of silicosis as an industrial disease, says that while to Collis belongs the credit of drawing attention to the effects of the inhalation of occupational dust containing free silica, even he did not visualize the widespread nature of the industries in which dust containing free silica occurs, as indicated by the recent plan for compensation under the Various Industries (Silicosis) Scheme, which covers such widely different industries as the manufacture of pottery, the making of scouring powders containing silica flour, the sandblasting of castings, the making of poultry grit by grinding and crushing of flints, the mining of tin and coal, and the making of vitreous enamels, as well as the sandstone industry affecting quarriers and masons.

One of the points emphasized by Bridge is that there are few, if any, dusts occurring in industry that will not produce, in certain individuals, an inflammatory condition of the skin; but that however distressing and economically wasteful simple lesions of the skin by practically all dusts may be, they are as nothing compared with cancer of the skin produced by dust. He states that although the main incidence occurs among the patent fuel workers in South Wales, the pitch workers in gas works, the loaders and unloaders of pitch, and the workers exposed to pitch dust in tar distilleries, all workers wherever they may be share the risk.

According to The Colliery Guardian,[9] a subject of increasing importance to the mining community is the occurrence of cases of silicosis in coal mines. Birmingham University hopes shortly to commence systematic chemical investigation of the shales in the vicinity of coal seams and those in which drilling and blasting have frequently to be done. It is also proposed to investigate the quantity and nature of the dust suspended in the air in underground places where drilling is being carried out.

It is the understanding of The Colliery Guardian[10] that the Home Office proposes to make certain alterations in the Various Industries (Silicosis) Scheme, 1928, that may especially affect coal miners. It was this cheme that the British Medical Journal characterized as an example of the unusual course taken by the Government of legislating first and postponing inquiry until a later date. The same paper emphasized the necessity for some standard of what constitutes an injurious amount of silica-dust. Not long ago the Trades' Union Council,[11] on behalf of the Miners' Federation, made representations to the Home Secretary that great hardship was being caused to workmen, especially in coal mines, by reason of the words "or any rock containing less than 50 per cent free silica" in the definition of silica rock in the scheme. These words were specifically inserted on the advice of the Geological Survey for the purpose of excluding the granite industry, pending further medical inquiry into the occurrence of silicosis in that industry. That inquiry, which has now been

8 Bridge, J. C., Remarks on Occupational Dust: British Medical Jour., No. 3598, December 21, 1929, pp. 1143-1147.

9 The Colliery Guardian, Mining Research at Birmingham University: Vol. 141, No. 3638, July 11, 1930, pp. 155-157.

10 The Colliery Guardian, Silicosis in Coal Mines: Vol. 141, No. 3633, August 15, 1930, pp. 591-592.

11 Ibid.

completed, found that silicosis in the granite industry is confined to certain occupations and, therefore, it is held that the reason for the adoption of the reduction as to 50 per cent has disappeared. The chief complaint of the Miners' Federation was that a heavy burden was imposed upon the miner by the requirement that he must prove that he had, within the previous three years, been drilling or handling one of the classes of stone included in the definition of "silica rock."

The Colliery Guardian considers that the main difficulty arising from the amended scheme will be in adjudicating upon claims by workmen who suggest that they have contracted silicosis through working in or traveling in roadways where stone-dust is used in accordance with the regulations to prevent coal-dust explosions. It is now provided that no dust shall be used for this purpose that may be prohibited by the Board on the ground that it would be injurious to the health of persons working in the mine. It is not thought that this regulation will prevent claims from being put forward by workmen, who may suggest that the contraction of phthisis is in itself the best proof that the dust employed is of a dangerous character.

The Workmen's Compensation Act, 1925, was extended to industries involving exposure to asbestos-dust, as a result of the report of an inquiry submitted to Parliament in March, 1930,[12] which stated that there was grave danger of workers in the asbestos industry contracting fibrosis of the lungs, culminating not merely in disablement but in death. Fourteen deaths were recorded and the disease affected one in eight of those employed, a figure that only applied to those who were actually in employment at the time the examination took place - roughly 2,000 people. The employers, who believed that the provisions of this bill were on the right lines, and the Trade Union Congress representing the industries and workers concerned, were also agreed.[13]

The Lancet[14] calls attention to the difference in the fibrosis resulting from asbestos-dust and that from silica-dust encountered in gold mines. The whorled formations seen in gold miners' phthisis are absent and are replaced by a diffuse fine interstitial pneumonia. The difference is also reflected in radiograms of the two conditions, the gold-miner's lung showing scattered dense rounded opacities which contrast with the fine basal mottling and linear striae which are characteristic of the asbestos-worker's lung. A peculiar feature of pulmonary asbestosis is the presence of the so-called asbestosis bodies. These were discovered by Prof. Matthew Stewart in the sputum of workers suffering from the disease; they have been demonstrated by a number of investigators in sections from the lungs and in lung juice removed during life by exploratory puncture or expressed at autopsy. The asbestosis body, which bears a curious resemblance to a minute crustacean, consists of a

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12 Merewether, E. R. A., and Price, C. W., Report on Effects of Asbestos Dust on the Lungs: Home Office, London, 1930, 34 pp.

13 The Lancet, Workmen's Compensation (Silicosis Bill): Vol. 2, No. 3, July 19, 1930, p. 163.

14 The Lancet, Pulmonary Asbestosis: Vol. 50, No. 13, March 29, 1930, pp. 701-702.

central core of asbestos fiber which appears to be embedded in a series of disks. One end of the fiber is often buried in a bulbous extremity, and this with the succeeding disks produces the appearance of a minute screw. The nature of the aggregation around the central spicule is unknown, but Gloyne has been able to demonstrate the central fiber by exposing the containing body to the action of sulphuric acid.

France.- The interest in France in the subject of the health hazard from dust is shown by the decree, issued by the French Ministry of Labor in November, 1929, which obliges doctors to make notification of pulmonary affections caused by dusts of silica, lime, clay, or coal, in order to obtain information upon which to base an extension of compensation for disease. Feil[15] is of the opinion that the decree should be modified to require notification of silicosis only, since other dusts, such as coal, iron, or clay, are injurious only when associated with free silica, which causes silicosis when inhaled.

Germany.- German investigators[16] consider occupational dust to be extraordinarily significant for industrial hygiene and to continue to-day to take first place among the causes of occupational diseases, especially tuberculosis of the lungs. Jötten-Münster[17] says that it is an outstanding fact, known for many years and from many statistics, that workers engaged in dusty trades have a larger death rate for tuberculosis of the lungs than other workers. He calls attention to the fact that the metal grinders in Solingen and Sheffield always die of phthisis at an age before their children are grown. In the Solingen district, for example, the death rate is double that of the rest of the population and the illness figure is four or five times higher. The file cutters and grinders show the highest mortality from tuberculosis. The workers in the metal mines do not have as many fatalities from dust-lung tuberculosis. In 1920, Ickert found a tuberculosis mortality rate of 19.5 per 10,000 among the bituminous marl-slate miners of the Mansfield mountain district where some years later Redeker found the rate to be 36.5 per 10,000 miners. As is well known, with stoneworkers four-fifths of all cases of death can be traced back to tuberculosis. This is especially true for the sandstone and flint workers, as well as for those who work in ceramic factories. Those who work in granite are somewhat less endangered, since it contains less crystalline SiO_2 than sandstone which consists principally of silica.

Jötten-Münster[18] describes the investigations carried out by the Government Research Division, of the Institute of Hygiene, University of Munster, Bochum, Germany, as follows:

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15 Feil, A., Contribution a l'étude des pneumokonioses professionnelles: Presse Médicale, No. 56, July 12, 1930, pp. 939-942.
16 Die gewerbliche Staublungenerkrankung: Beihefte Zentralb. Gewerbeh. Unfallverhütung, No. 15, pub. by Julius Springer, 1929, 146 pages.
17 Jötten-Münster, K. W., Staublunge und Staublungentuberkulose: Beihefte Zeutralb. Gewerbeh, Unfallverhütung, No. 15, pub. by Julius Springer, 1929, pp. 22-49.
18 See footnote 17.

In our first series of experiments we took up the notoriously injurious steel grinding dust and the apparently harmless lime dust in order to compare the action of these dusts with that of the disputed dusts of porcelain, coal, and the closely related lampblack.

In the second series of experiments, the certainly injuriously active clay-slate (argillite) dust (high SiO_2 content) was compared with the apparently harmless cement dust and the questionable tobacco dust.

In the third series, finally, we experimented with the limestone dust present in the harmless cement, the noninjurious Thomas slag dust, and the questionable cotton textile dust.

Rabbits were used for experimental animals, after we had determined previously that experimental rabbit tuberculosis produces tuberculous lung processes, which resemble human lung tuberculosis from the standpoint of running a longer-lasting course much more than the tuberculosis that can be caused experimentally in guinea pigs.

Before beginning the dust experiments, we succeeded through a very slightly fixed infection brought about by means of inhalation of weak rabbit-virulent and human bacilli in causing an immunity-trough-disease in the experimental animals as we have done with grown, already tuberculous-diseased human beings. The secondary infection renewed later toward the close of the dusting lead to a secondary, that is, chronic indurative tuberculosis. This is a new investigative possibility which has not been used up to this time with tuberculosis experiments.

Jötten-Münster summarizes the results of these experiments as follows:

The severest changes were, as was to be expected, caused by the steel grinding dust, the next by the dusts with high silica content, such as rock dust (with clay slate), and porcelain. Of these latter, the higher the silica content the more injurious they were. Coal dust and lampblack had a moderately injurious effect, while cement, limestone, Thomas slag, textile, and tobacco dust were still less injurious, with lime dust in the form of $Ca(OH)_2$ the least harmful.

On the basis of the experiments, the disputed porcelain dust as well as the silicates can in no way be considered as harmless. Also, it is not claimed that anthracite coal dust is completely indifferent or checks tuberculosis entirely; it stands close to the zone of indifference; it even promotes tuberculosis. The tubercle has developed everywhere around the coal particles. The opinion is correct that the relative infrequency of tuberculosis among coal miners is due not to a tuberculocidal property of the coal dust but much more to the careful occupational selection, possibility of earlier disability of the miners, and other exceptional elements. The reports that the condition of other coal-dust workers, for example the men unloading the coal, does not remain for long as favorable as that of the miners indicate this.

What has been said for anthracite coal dust applies also to cement and limestone dust, which experimentally, indeed, are less injurious than coal dust but are not entirely indifferent. Results of investigations in the industry itself strengthen these views. The animals exposed for four months in the packing room to about 20 mg. dust in a cubic meter of air, in the cement mill to about 160 mg. of dust in a cubic meter of air, and those in the ball mill with about 300 mg. of dust per cubic meter of air showed the same reaction to dust and a slight promotion of tuberculosis, when infected by tubercle bacilli, as did the laboratory animals.

Our results with lime dust were especially favorable. The harmlessness of the lime may be explained principally by the fact that it dissolves in the tissue liquids and, therefore, exerts no injurious mechanical action. In addition the inhalation of lime assists the calcification of the tuberculous foci and in this way the tubercle bacilli are immobilized.

There seems to be some difference of opinion among German investigators on this subject of the effects of dust. Hollman,[19] of Solingen, makes the following deductions from his studies:

> Tuberculosis, which today in industrial regions accompanies men from the cradle to the grave, we need not regard as an occupational disease, as the course of the disease is not more severe, the fatality not higher among grinders than among other industrial workers. Infection and reinfection are decisive for its progress which depends upon exposure and the ability to withstand contagion, and these are not unfavorably affected by occupation.
>
> There is even discussion that with slight production of grinding dust the resistance to infection can be increased, that these cases run a milder course, conforming to the Arndt-Schulz law of the effect of weak irritation.
>
> There is an individual constitutional disposition toward pneumoconiosis, the preliminary conditions of which have not been satisfactorily investigated in a scientific manner. It appears in our metal grinders no more frequently comparatively than with workers in other occupations and, on the other hand, does not parallel dust production.
>
> For the prevention of pneumoconiosis as a disease limiting industry and threatening life, X-ray examinations should have place under the executives of the dusty industries, to which metal grinders belong. Such investigations, that is with good apparatus, are sufficient every two or three years.

19 Hollman, Schleiferkrankheit und Lungentuberkulose: Beihefte Zentralb. Gewerbeh. Unfallverhütung, No. 15, 1929, pp. 128-141.

The reduction of deaths among grinders in Solingen during the last 30 years - according to Teleky from 1923 to 1927, 40 per cent for Kronenberg and 30 per cent for Remschied - encourages us to strive for the goal by the two attainable methods of reducing the dust hazard and reducing the operation of tuberculosis infection, which brings up the most important problem of medical activity, prophylaxis.

Von Döhren[20] also mentions individual disposition toward the development of pneumoconioses. He says that whether this individual disposition is postulated by an insufficiency of the reticulo-endothelial apparatus, which is perhaps hereditary, has not yet been indisputably determined. Although some authors write that in the history of the dust-lung patients statements are found frequently regarding recovery from lung diseases, von Döhren could not confirm this observation by his material. Regarding the relations between tuberculosis and pneumoconiosis, he found that among 217 dust lungs, 6.91 per cent were associated with tuberculosis. The following figures are given regarding the frequency of pneumoconiosis among rock-drillers in the Rheinisch-Westphalic coal district: Among 11,427 miners of all occupations, 3.12 per cent were found to have pneumoconiosis, 24.4 per cent of the cases being associated with tuberculosis. The pneumocotic changes were determined among rock-drillers only, and not among purely coal-drillers, timbercutters, or truckers. Among 1,000 drillers who had worked previously in rock, 217 had dust lungs, or 21.7 per cent.

According to von Döhren, the time required for the development of pneumocotic changes is extremely varied. Observations of entirely isolated cases, in which after three to four years severe changes with or without tuberculosis were noticeable, show that very few become ill during the first five years. With increasing length of working time, however, the percentage goes up by leaps and bounds.

According to Sternberg[21] there is still a wide gulf between the findings of industrial dust lung and the results of animal experiments. No one has yet been able to produce by animal experimentation such a fibrosis as that shown by a rock-driller's lungs. He advances the following theories to explain this variation:

The anatomical-physical difference between human lungs and animal lungs, which began as a result of the upright posture.

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20 von Döhren, Zur Klinik der Staublunge: Beihefte Zentralb. Gewerbeh. Unfallverhütung, No. 15, 1929, 146 pp.

21 Sternberg, M., Die Staublunge: Med. Klin., vol. 25, No. 50, December, 1929, pp. 1919-1922.

The fibrous dust lung of human beings requires several years, even ten years, for development. A similar period of time is not possible in animal experiments due to the shorter life span of experimental animals.

Schridde has found the structure of the rock-driller's nodules to be identical with the keloid of the membrance and is of the opinion that a special predisposition is necessary for the genesis of lung keloid.

The bacterial theory assumes that the genesis of a genuine fibrosis depends upon the assistance of bacterial infection of the lungs in addition to the action of the dust. This pushes to the fore the question of dust tuberculosis, which is the most frequent and the most usual bacterial infection of the lungs.

It is certain that there are cases of silicosis and indurative anthracosis, even with cavity formation, in which the most careful microscopic examination reveals not a trace of tuberculosis. Genuine lung fibrosis can develop in people without tuberculosis. However, there are regions and occupations in which practically all workers in dust who have lung fibrosis at the same time are tuberculous.

Grünewald[22] believes that the worker in the dust-producing industries seldom pays any attention to the first results of the action of dust but believes the slight cough, sneeze, and catarrh as necessary evils of his occupation or as inherited weaknesses of his family. Thus frequently from simple catarrh of the nose or throat come deep-seated changes of the lungs, such as swelling, chronic inflammation, induration, and phthisis. Grünewald states that the significance of dry dust for the origin of an infectious disease, as for example tuberculosis, is really not that of the infection carrier but of a physical and chemically active preparatory irritation for retaining the tuberculous infection. In regard to coal-dust he thinks that

> Of all dust diseases of the lungs, anthracosis is the least injurious, although at times, as a result of longer action of the coal dust, lung and air passages are distended. A special tendency of the miner's lungs toward tuberculosis is certainly to be denied. However, if the coal dust has been absorbed before breathing tubercle bacilli it is as dangerous as any other dust that contains tubercle bacilli.

Grünewald calls attention to the order of the Labor Minister of February 11, 1929, making compensable dust-lung diseases contracted in certain industries, although the order of May 12, 1925, did not include illness caused by dust.

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22 Grünewald, Max, Die Berücksichtigung der Krankheiten der Atmungsorgane durch Staub und Gas bei der Ausdehnung der Unfallversicherung auf Gewerbliche Berufskrankheiten: Die Gasmaske, vol. 1, No. 6, December, 1929, pp. 145-148.

Russia.- Investigations on various phases of the dust hazard were conducted by the State Institute for Occupational Pathology and Industrial Hygiene of Ukraine.[23] Studies[24] on the relation between pneumoconiosis and tuberculosis consisted in experiments on animals, clinical examination of the personnel of the State Porcelain and Faience Factory, and a statistical study of mortality from tuberculosis of these workers during a 12-year period. The experimental study consisted in determining the effect of injections of colloidal solutions of SiO_2 under the skin of rabbits infected with tuberculosis and those not so infected, and dusting the animals with faience dust in order to determine the local action on the tissues. Comparison of the data obtained on the experimental animals with that on the controls showed that SiO_2 promoted the growth of the fibroblastic element so that in a series of cases tuberculous foci were incapsulated; the tuberculous process, however, progressed and showed no tendency to change a malignant course into a mild one. Similarly a causal connection between fibrosis of the lungs and the presence of a very marked destructive process was not determined. The clinical examination of the factory workers showed that the percentage of silicosis and tuberculosis was very high but the form of compensated tuberculosis predominated and only at a certain age (45 to 50 years) after a long period of work (15 to 20 years) did a "secondary wave" of exacerbation of the tuberculous process occur. This secondary exacerbation is explained by the author as due to the presence by this time of too much connective tissue in the lungs, which leads to vascular injury.

Studies carried out to determine the parthological effects[25] of sand, flint (silicon), porcelain, chamotte, anthracite, coke, charcoal gypsum, and marble dust gave the following results: Porcelain-dust, was easily hydrolized and, when introduced into the animal organism caused local necrotic lesions and toxic effect of the internal organs; with rabbits infected with tuberculosis the tuberculous process was localized in the region of "dust foci" which referred back to the necrotic action of this dust on the tissues; crystalline kinds of silicate dust (flint, sand) were not hydrolized; schamotte stood between porcelain-dust and flint and sand as far as its hydrolytic properties were concerned; coal-dust was not soluble in the organism; tissue reaction with dusts proceeded with definite activity of the machrophages; with infected animals no localization of tuberculous infection took place around the "foci" with coal-dust. Charcoal-dust showed no noticeable picture of solubility and tissue reaction; marble and gypsum dusts are hydrolyzed in the organism, and tuberculous lesions can be determined around the foci of these dusts.

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23 Staub und Staubpathologie: Arbeiten und Materialien des Ukrainischen Staatsinstituts für Arbeitspathologie und Arbeitshygiene, Kharkov, Russia, 1930, 292 pp.

24 Sheinin, M., Die Silicose und Tuberkulose im Lichte der klinischen und experimentellen Untersuchungen: (German summary of Russian report) Arbeiten und Materialien des Ukrainischen Staatsinstituts für Arbeitspathologie und Arbeitshygiene, Kharkov, Russia, 1930, 292 pp.

25 Peissachovitsch, I. M., Pathologie des Staubes: Arbeiten und Materialien des Urkainischen Staatsinstituts für Arbeitspathologie und Arbeitshygiene, Kharkov, Russia, 1930, 292 pp.

South Africa.- According to the report of the Miners' Phthisis Medical Bureau,[26] 424 of the 15,492 working miners examined during the year 1928-1929 had simple silicosis. Of these 424 cases, 154 were "old" cases of silicosis, made up of miners who had previously been certified as having simple silicosis, but who had not retired from underground work. Of these, 106 were in the ante-primary, 57 in the primary, and 1 in the secondary stage of the disease; 112 were mine officials. The remaining 270 cases were "new" cases of silicosis which had originated during the year. During the year five new cases of tuberculosis with silicosis were found among the 15,492 working miners examined. All these cases occurred among unretired miners who had previously been notified as having simple silicosis. Not one was a "new Rand miner" and none had at any time passed the initial examination of the bureau. Two were "old Rand miners," one with 8 years and 11 months' and the second with 12 years and 6 months' underground service; neither had worked machine drills. Three belonged to the class "miners, Rand and elsewhere," Their average period of work in scheduled mines was 14 years and 9 months, with 7 years and 2 months elsewhere. Two had done some machine work but neither had been exclusively so engaged. Forty-four of the 15,492 miners examined were found to be suffering from "simple tuberculosis." Four of the cases, two in the "open" and two in the "closed" form, were among the "new Rand miners." The mean duration of underground service for this class prior to the contraction of tuberculosis was 6 years and 2 months; 35 of the cases, 29 "open" and 6 "closed," were among the "old Rand miners." The mean duration of the underground work of the 35 cases was 12 years. Five cases of "open" tuberculosis were found among those classes as "miners, Rand and elsewhere." The mean duration of underground work in these cases was 11 years and 9 months in scheduled mines with 10 years and 9 months' underground work elsewhere. Four had worked on machine drills but none had been exclusively so engaged.

According to the report, detailed data regarding the subsequent history of each annual group of the new cases of tuberculosis with silicosis and of simple tuberculosis detected at the periodical examinations since 1917-1918, show plainly that the liability to death bears a direct relation to the relative prominence of the tuberculous factor in each class of case. It is least in the group of originally "ante-primary" cases of simple silicosis; somewhat higher in the group of originally "primary" stage cases; higher again in the cases of "closed" tuberculosis - usually of the chronic fibroid type - which form a link between the cases of originally uncomplicated silicosis and the two classes in which an active tuberculosis is present from the outset. The two latter exhibit the highest mortality, and an early conjunction of the obvious active tuberculosis with silicosis produces a condition that is more rapidly fatal than an active simple tuberculosis alone. The association of an infective factor, and particularly of tuberculous infection, with the progressive tendency manifest in most cases even of simple silicosis has been repeatedly emphasized in these reports.

26 Irvine, Louis G., Report upon the Work of the Miners' Phthisis Medical Bureau for the Year Ended July 31, 1929: Government Printer, Pretoria, South Africa, 1930, 64 pp.

In regard to the position today of miners' phthisis on the Witwatersrand, Irvine and Mavrogordato[27] have the following to say:

> Although one is justified in concluding that we have turned a big corner in the matter of silicosis, it is not anticipated that the actual number of cases detected will show any significant further decrease in the immediate future. For, although the "New Rand Miners," with their relatively lower attack-rates, are becoming more numerous and the older miners fewer, nevertheless, the older miners who remain - and there are still over 5,000 of them - are every year getting older in years of service, and therefore more liable to contract silicosis. The one factor will probably for some time balance the other, and one does not therefore anticipate much, if any, actual decrease in the number of cases that will arise for a good many years to come.
>
> That is the position today. After years of hard work on the part of mining engineers and inspectors and mine officials and medical men, a large measure of success has been attained. But it is incomplete. It is satisfactory so far, but it is also disappointing. When in 1916 we found that the liberal use of water, suitably applied, would reduce the dust from drilling or blasting by 97 or 98 per cent by weight, the problem appeared to have been solved. It has only been solved <u>so far</u>. The trouble is that, although water will take one a large part of the way, it will not take one all the way. It has its limitations, and it has also positive disadvantages, both hygienic and economic. Hence the minds of engineers and medical men are turning today to the question: Have we not been over-doing water? Could we not do better with less water and a greater extension of alternative methods?

The matter of compensation for miners' phthisis is causing considerable concern to the industry in South Africa, as indicated by the following statements from the South African Mining and Engineering Journal:[28]

> The outstanding liabilities of the mines of the group to the Miners' Phthisis Board at 31st July, 1929, amounted to £3,150,746, as compared with £3,072,439 the year before. All mines that are in a position to do so are setting aside funds annually, so that the whole amount of their respective liabilities will have been provided during the course of their industrial lives. ***********
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> In addition to the above, the amounts contributed out of working costs by the companies of the Group to the compensation fund of the Phthisis Board amounted in 1929 to £385,729, equal to 4s. 8d. per underground shift. If the amounts provided during the year to meet the outstanding liabilities are added to the contributions to the compensation fund, the total cost to the mines of the Group on account of miners' phthisis represents the equivalent of 6s. 10d. per shift for each white employee working underground.

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27 Irvine, L. G., and Mavrogordato, A., Miners' Phthisis on the Witwatersrand: South African Min. and Eng. Jour., April, 1930, p. 73.

28 South African Mining and Engineering Journal. The Cost of Miners' Phthisis: May 10, 1930, page 283.
South African Mining and Engineering Journal. Miners' Phthisis Legislation: August 2, 1930, p. 650.

The report of the latest Miners' Phthisis Commission is now due and the moment may be opportune to review the attitude of the mining industry toward this much-debated subject. That attitude has been very clearlv explained by the industry in the evidence given on its behalf before the Commission. On behalf of the industry it was admitted that there should be permanent and lasting legislation on this contentious subject; but it was emphasized that to frame such legislation on retrospective lines and to provide for so-called "hard cases" is bound to produce unsatisfactory results. There are at present 35 schedules mines which are liable to contribute to the annual levy of £800,000 and provide for the net outstanding liability of £6,400,000. Several of these mines have, as yet, not been in a position to set aside annually against their outstanding liability to the Compensation Fund. Although two mines closed down during the year 1929, the Board has maintained the annual levy at £800,000, owing to the large outstanding liabilities of the Compensation Fund; so that these contributions have fallen with cumulative effect upon the other scheduled mines. *********

If additional benefits for miners' phthisis sufferers and their dependents, are recommended, such additional benefits should be borne by the State; in fact, it is urged by the gold producers that it would even be in the interests of the State at the present time to assist the gold mining industry in meeting some of its obligations under existing legislation. *********

Owing to the great amount of attention which has been given to miners' phthisis and the generaous scales of compensation provided, there is evidence of the growth of an idea among a section of mine workers that after they have worked a certain number of years underground, they should be provided with a pension for life, whether they be sick or sound. This is, of course, a sorry state of affairs in view of the strenuous and successful efforts made to improve conditions and the huge present outstanding liabilities placed upon the mines. It suggests the idea that a gold-mine is a bottomless purse and that one has only to ask in order to get. This attitude, one hopes, for the sake of the country as well as of the industry, will not be allowed to develop.

In August, 1930, an international conference on silicosis was held at Johannesburg, South Africa, which was attended by representatives of the principal countries interested in this industrial disease. According to a summary of the results of the conference by Hall[29] no material addition was made to pre-existing knowledge but it served a useful purpose in many ways.

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29 United States Department of Labor, Proceedings of the Sixteenth Annual Meeting of the International Association of Industrial Accident Boards and Commissions: Bureau of Labor Statistics Bulletin 511, April, 1930, pp. 40-47.

For instance, by collecting together all the available information about dust diseases from all parts of the world, it has shown not only the number but also the position of the gaps in our knowledge of the subject, and so has given valuable indications for future lines of research. It has enabled certain standards to be agreed upon, by which future work on this subject may be comparable in whatever country or industry it is carried on. Finally, it has emphasized the urgent necessity for further experimental investigations into almost every aspect of the problem.

United States.- In April, 1929, a committee was appointed by the industrial commissioner, of New York to draft, for recommendation to the industrial board, rules relating to the regulation of rock drilling, sand blasting, and rock crushing. A report of this committee was made by Lanza[29a] at the session on occupational diseases of the Sixteenth Annual Meeting of the International Association of Industrial Accident Boards and Commissions, Buffalo, New York, October 8-11, 1929. It had been thought that the problem was that of the application of water to rock drilling, but time showed that this was not the solution as the fine dust particles came up through the water and got into the atmosphere just the same and caused silicosis as they had previously. In Manhattan it was obvious that on many of the operations - foundation work, tunneling, and subway work - especially in the wintertime water could not be used because it would form into ice and probably increase very considerably the accident hazard. Therefore, the State department of labor concentrated its attention on developing some other method of dealing with the dust problem. In its efforts, the department was assisted by the contractors, who tried to evolve some sort of a suction apparatus that would take the dust away as rapidly as it was formed at the drill hole. However, this plan proved to be not only difficult but relatively quite expensive to execute. Such a device was tested in actual operation at a subway tunnel construction. First, before any work was done the air was sampled to determine the normal dust content of the air at that place, which was found to be slightly more than 23,000 particles per cubic foot - that is, particles under 10 microns in size. Four jack hammers were then put in operation with the dust collecting device working, and the count jumped from 23,000 to 114,000 particles per cubic foot. The dust-suction apparatus was then removed and the count jumped from 114,000 to 423,900,000 per cubic foot. This was very encouraging, as the United States Public Health Service has adopted as a permissible limit 10,000,000 particles under 10 microns per cubic foot. Although it can not be said that the problem has been solved, Lanza stated they considered the most hopeful, the most promising experiment yet made. During the discussion of Lanza's report, the question was asked as to the time required to develop silicosis after exposure to silicate dust. He replied that a man working steadily in silica-dust running 99 per cent plus of silica will be seriously and usually permanently damaged at the end of five or six years; if in silica dust containing 70, 65, or 60 per cent silicate, with perhaps little better working conditions - and that is about the average in this country, with one or two notable exceptions - the time required will run to 10 to 15 years. It is a slowly developing disease and under most working conditions in this country it is a matter of 10 to 12, 14 or 15 years before a man is aware, providing he has not been medically supervised, that he is thoroughly damaged.

29a See footnote 29.

It was also brought out in the above discussion that in Ontario, Canada, the law provides that a man must have been exposed for five years in one of the Ontario mines before he becomes entitled to the silicosis provision. Every man is examined before he is allowed to go to work, and if he has had experience in working in silica dust and has silica on his lungs, he is refused employment. No man with a suspiciion of silicosis is allowed to go into the mines of Ontario.

The work of the U. S. Bureau of Mines Clinic, carried on at Picher, Okla., in cooperation with the Tri-State Zinc and Lead Ore Producers Association and the Metropolitan Life Insurance Co., was continued during the fiscal year 1929-30. During this period, 7,469 men were examined, of which number 3,002 were original (first annual) examinations, and 4,457 were re-examinations (second or third annual). There was a slight reduction in the number of men examined compared with the number examined last year, due to the financial depression existing in the district. A complete tabulation of the physical condition of those examined is being prepared for possible publication.

An interesting development in connection with the investigation conducted at Picher was the discovery by X-ray examination of a miliary lung disease of unknown origin[30] in 125 of the 18,285 individuals examined up to December 31, 1929. With one exception, these were all white, native Americans, the majority of whom were born and reared in the vicinity of the mining field. Most of them came from rural districts or had spent several summers in the harvest fields. A majority of these cases did not have sufficient symptoms to cause them to stop work or to seek medical aid. Many of them were men seeking employment or who had been employed underground for only a short time at most; and, apparently, mining work could not have caused the condition found. In attempting to make a diagnosis, four diseases - miliary tuberculosis, pneumoconiosis, calcium metastasis, and pneumomycosis - were considered as probable causes of the condition. However, the investigation at Picher, as well as a search of the literature, failed to reveal sufficient knowledge of this malady to warrant a statement that the cases under discussion come under one of the four probable diseases mentioned, but there is enough evidence to conclude that the disease is more prevalent than thought and for this reason is worthy of serious study.

Another dust disease, asbestosis, has assumed a prominent and perhaps peculiar interest in the field of pneumoconiosis and occupational disease.[31] Although the asbestos industry is more than 2,000 years old, it is only within the last few years that it has become so important. Until recently asbestosis was assumed to be essentially silicosis, and the free silica has been thought to constitute the dangerous factor. However, according to Lynch and Smith,[32]

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30 Sayers, R. R., and Meriwether, F. V., Miliary Lung Disease Due to Unknown Cause: Public Health reports, vol. 45, No. 49, December 5, 1930, pp. 2994-3009.

31 Lynch, K. M., and Smith, W. A., Asbestosis Bodies in Sputum and Lung: Jour. Am. Med. Assoc., vol. 95, No. 9, August 30, 1930, 659-661.

32 Ibid.

attention has been called within the last two or three years to a peculiar characteristic of asbestosis, not found in silicosis, and on which they reported data, as American medicine had thus far had no report on this peculiar condition. As mentioned above,[33] an investigation was conducted in England prior to which (February, 1928) definite knowledge existed of only two deaths of asbestos workers about whom there was expert opinion that the inhalation of asbestos-dust had at least contributed to, if not caused, the fatal outcome.[34] The second of these cases, reported by Cooke,[35] in 1924, was that of an asbestos worker who had been exposed for 20 years, the last 5 years intermittently, and who on autopsy showed extensive fibrosis of the lungs and chronic tuberculosis. A histologic study by McDonald[36] of the lung tissue of this case showed in addition certain peculiar foreign bodies in the alveoli, bronchioles, and interstitial fibrotic areas. This is the first recorded observation of these objects which since have been called "asbestosis bodies." In October, 1929, Lynch and Smith[37] found these bodies in two negroes coming to autopsy, one of whom had died of a gunshot wound and the other of lobar pneumonia. One of the men had worked in an asbestos mill for a total of 28 months during a period of about three years. The other had been working in an asbestos mill almost continuously for four and a half years. It then occurred to Lynch and Smith that these "bodies" should be found in the sputum of asbestos workers. The "asbestosis bodies" were found in the sputum of three of four patients who had been or were working in asbestos. The case in which no "asbestosis bodies" were found had worked for 14 years in an asbestos factory but not since 1926. He had advanced fibrosis of the lungs, thought to be a late stage of pneumoconiosis, without evidence of tuberculosis. The relation of the occurrence and time of duration of these bodies in the sputum to the extent of exposure to asbestos dust, and to the state or stage of asbestosis or other associated conditions in relation to their expulsion in the sputum should be given further study in the opinion of Lynch and Smith. Meriwether[38] considers the following four conditions essential for establishing a relationship between the inhalation of asbestos-dust and the development of fibrosis:

1. Work involving exposure to asbestos dust.

2. The existence, demonstrable clinically and radiologically, of a definite pulmonary fibrosis.

3. The absence of previous or present infections known to cause pulmonary fibrosis - e.g., tuberculosis, influenza, or pneumonia.

4. The absence of previous or present work involving exposure to other dusts, which might cause pulmonary fibrosis.

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33 See footnote 12.

34 Merewether, E. R. A., The Occurrence of Pulmonary Fibrosis and Other Pulmonary Affections in Asbestos Workers, The Jour. of Indust. Hyg., vol. 12, No. 5, May, 1930, p. 198-222.

35 See footnote 31.

36 Ibid.

37 Ibid.

38 See footnote 34.

Soper, in describing a case of asbestosis,[39] is of the opinion that the condition is more widespread than at present suspected and that just as certain other dusts took their toll over a long period before the true facts were realized, so asbestos-dust to a lesser degree appears to be taking its toll without sufficient recognition of the fact. He makes the following statements in regard to the disease:

1. A case is reported which seems to be typical of pulmonary asbestosis.

2. When there has been exposure to asbestos dust the presence of pulmonary asbestosis should always be suspected.

3. The most common single symptom is dyspnoea. This and the other symptoms are essentially those of a progressive generalized lung fibrosis.

4. The physical signs of uncomplicated pulmonary asbestosis are substantially those of generalized fibrosis of both lungs and basal pleurisy.

5. X-ray examination is of great value in establishing the diagnosis.

6. Asbestos contains but a very small amount of free silica but probably conduces to a more hasty evolution of any accompanying tuberculosis, as in the better understood forms of silicosis.

7. An immediate diagnosis at autopsy is said to be made possible by simply squeezing out upon a slide a drop of lung juice from the fibrosed tissue and covering with a cover-glass. The asbestosis bodies in large number are readily visible under the microscope.

Hatch, Drinker, and Choate[40] made a study to determine the fundamental engineering principles involved in the design of efficient dust-control systems for use with high-speed pneumatic stone-cutting tools. They summarize the results of their laboratory study as follows:

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39 Soper, W. B., Pulmonary Asbestosis; The American Review of Tuberculosis, vol. 22, No. 6, December, 1930, pp. 571-583.

40 Hatch, T., Drinker, P., and Choate, Sarah P., Control of the Silicosis Hazard in the Hard Rock Industries. I. A Laboratory Study of the Design of Dust Control Systems for Use with Pneumatic Granite Cutting Tools; Jour. Indust, Hygiene, vol. 12, No. 3, March, 1930, pp. 75-91.

Of the available methods of dust control, the application of local exhaust ventilation appeared to be the most promising. A study of the various granite cutting processes showed that the surfacing machine and the pneumatic hand tool alone needed investigation, since they along produced amounts of dust in excess of the standard. The heaviest dust producing tool, the four-point, was used throughout the study.

Four different hood shapes for use with the pneumatic hand tool were tested and the minimum air velocity required at the tool to keep the dust concentration at ten million particles was determined. When the hoods were operated at the minimum rates of air flow, the velocity at the tool was found to be 200 feet per minute, regardless of the size, shape, and position of the hood and the air flow through it. This value, therefore, may be regarded as a fundamental specification for hood design under laboratory conditions. For the field, it may have to be altered somewhat.

The hood shape finally adopted for use with the surfacing machine encloses the process as completely as possible. Physical barriers arrest the dust thrown away from the hood and high air velocities are created at strategic points without increasing the total air flow. The air flow requirement for the smaller surfacing machine was found to be 315 cubic feet per minute. A higher rate will be necessary when the larger machine is used.

SUMMARY OR RECENT LITERATURE ON EFFECTS ON WORKERS OF EXPOSURE TO TOXIC OR NOXIOUS GASES

Next to dust, the most important hazard to health through lack of proper ventilation is the possible presence of toxic or noxious gases in the air breathed. This hazard is present everywhere, even in the home, and is increasing with the rapid development of new chemical products. Following is a summary of some of the reports published on this subject in various industrial countries.

England.- The following table reports[41] cases of gassing for the year 1929, as compared with previous years. The figures relate to the number of cases reported, with the number of deaths in parentheses.

41 Bridge, T. C., Industrial Disease Among Chemical Workers: Chem. Age, vol. 23, No. 581, August 16, 1930, p. 139.

Comparison of data on gassing, 1925 - 1929

Agent	1925	1926	1927	1928	1929
Carbon monoxide............	118 (10)	101 (6)	88 (4)	81 (9)	113 (10)
(a) Blast furnace	25 (6)	9	13	22 (8)	25 (5)
(b) Power	34 (1)	32 (2)	19 (1)	20	55 (3)
(c) Coal	26 (2)	26 (1)	38 (2)	14	21 (1)
(d) Other	33 (1)	34 (3)	18 (1)	25 (1)	12 (1)
Carbon dioxide............	10 (2)	4	3	8 (1)	- -
Sulphuretted hydrogen	4	3	9	9 (3)	7 (2)
Sulphur dioxide	3 (1)	2	5 (1)	10	6
Chlorine	12	13	14	17	14
Nitrous fumes	10 (2)	5 (1)	7	6 (1)	11 (2)
Ammonia	5	5 (1)	5	12 (1)	18
Benzol, benzine, naphtha and petrol	3 (1)	4 (1)	7 (2)	7 (2)	7
Other (ether, acetone, nickel carbonyl, etc.)...	35 (3)	17 (1)	23 (2)	17 (2)	36 (1)

The Senior Medical Inspector[42] calls attention to the fact that the number of carbon monoxide cases was higher in 1929 than for previous years, the reason being due to cases which have occurred at a large works where water gas is used. Accidents from carbon monoxide are often very difficult to avoid, and while the number of fatalities is not high, the possibility of many nonfatal accidents becoming so is apparent. Accidents of this class can only be avoided by the greatest care and supervision. Workers must have impressed upon them the necessity of making every use of the appliances available - breathing apparatus, rescue lines, etc. "Familiarity breeds contempt," and employers should realise that workers, in order to carry out necessary work, often take risks when they would hesitate to do so if warned fully of the dangers. There is no accident more tragic than a fatal gassing by carbon monoxide, because almost invariably one has the feeling that with foresight it could have been prevented. Unfortunately, one man overcome by carbon monoxide gas in doing some simple job, jeopardises the lives of other men who attempt rescue. It is not sufficient to have appliances for rescue or resuscitation available; the workers should be instructed in their use. Every accident of this character, even slight cases, should be carefully inquired into by every person responsible and by those who actually do the work. In such a way a better sense of the danger of ordinary jobs would be established. If the results of such an inquiry had the fullest publicity, then those engaged on similar work might be in a position to anticipate and provide against similar accidents.

The five nonfatal sulphuretted hydrogen cases were due to escape of H_2S in gasworks; three of them occurred in emptying purifiers. The two fatal cases were of an entirely different character, and affected three men engaged on the enlargement of a cesspit on factory premises. The hydrogen sulphide gas accumulated during the night and arose from the decomposition of organic matter. No tests were made before descending into the pit in the morning, and the first man to descend was immediately overcome and fell from the bucket in which he was being lowered. Brave efforts were made at rescue, and in these attempts another man also lost his life.[43]

42 See footnote 41.
43 See footnote 41.

According to the British Medical Journal,[44] Sir Bernard Spilsbury, taking up the reference by Professor Haldane to the controversy as to whether carbon monoxide had any direct action upon the tissues apart from the changes in the blood, by reducing the oxygen-carrying capacity, said that it had long been known that in the more chronic forms of profound anemia very characteristic changes occurred in the body, such as an extremely fatty disease of the heart muscle and the kidneys. It was not always realized how rapidly such changes could occur in a relatively short period of anemia if the oxygen-carrying power of the blood were lowered. He had had the opportunity of investigating cases where the hemoglobin percentage had been considerably reduced for some time, and it only required a severe anemia of three or four days' duration to induce these fatty degenerations. If fairly tough organs like the liver and kidney could be so affected by anemia of that duration, certainly the brain cells must be affected; and thus the profound results of a prolonged oxygen deprivation could be accounted for. With regard to the dangers of chronic carbon monoxide poisoning to the modern community, there could be little doubt that the exhaust gases of motor vehicles would constitute a serious menace in the future. He had very little doubt that many cases of chronic CO poisoning had been overlooked in the past and present. It had occurred among workers in large garages where engines were running frequently. Unless the garages were well ventilated the possibility of CO poisoning was very present. In estimating the fatal saturations with carbon monoxide, Spilsbury said he had never obtained figures quite so high as Professor Haldane had suggested - namely, 80 per cent; he had occasionally found 75 or 76 per cent, but in most of the fatal cases he had investigated the figures were nearer 60 to 65 per cent, as an average. In very few cases had he found death occurring with a low percentage, and there could be little doubt that while a normal person required a very high percentage for a fatal dose, a person in ill health might die with a very much lower percentage in the blood. He recalled the case of a young woman who committed suicide in this way, and in her case the saturation was only 45 per cent. She had with her a child of 7, who was unconscious, but survived. The woman herself was suffering from advanced chronic tuberculosis of the lungs, and he thought the low fatal dose was occasioned by the fact that she would naturally offer less resistance than a healthy person. In two other cases, in which the percentage was about 50, one was an old and feeble person, and the other was suffering from cancer of the stomach. With regard to the recognition of carbon monoxide in the body after death, the post-mortem appearances were extremely characteristic, so that one could often diagnose a case of carbon monoxide poisoning on sight. But according to Spilsbury there was one condition that might mislead a medical man not familiar with the changes which occurred. A doctor, searching through the organs of the body, might find nothing at all to indicate the reason for death, and then be suddenly struck with the fact that the blood in the organs was bright red, so that he would jump to the conclusion that it was a case of coal gas poisoning. But that bright redness might be entirely due to the changes in the viscera after exposure to air. The blood in the viscera is of dark color when first exposed, but after exposure it becomes a bright red. One other form of

44 British Medical Journal, Carbon Monoxide Poisoning: No. 3626, July 5, 1930, pp. 17-18.

poisoning in which sometimes the blood and even the viscera might have rather a bright color is cyanide poisoning. This condition is almost the only one through which a mistake in diagnosis of the cause of death is likely to be made.

The medical school of Leeds University[45] is studying the effects of carbon monoxide poisoning on the nervous system by experiments on birds and small animals. It has been found that, with repeated exposure to small concentrations of gas, canaries develop a tolerance that renders them unreliable for detecting concentrations below about 0.2 per cent. Lung hemorrhages were found to occur in all birds subjected to three or more exposures.

France.- The Committee of the Prefecture of Police for the Suppression of Fumes and Noxious Automobile Exhaust Gas, appointed in 1927 to study the practicability of the suppression of such fumes and gas, has published its report.[46] It considered the problem as analogous to the suppression of industrial smoke, since fumes from both sources are the product of incomplete combustion of carbonaceous materials.

Tests were made to determine the amount of carbon monoxide produced by different makes of automobiles, the effect of antiknock substances, the diffusion of CO in the air from automobile exhaust gas at various points in the city of Paris, and the effect of the gas on vegetation. The investigation showed that motors could be so constructed as to emit only a small amount of toxic gas. The results of the analyses of the air in the streets was reassuring as regards any danger to health. Certain trees were found to be particularly susceptible to dust and exhaust gas, and it was recommended that they be replaced by more resistant species. Two measures were proposed to reduce to a minimum the inconvenience from the emission from automobiles of carbon monoxide as well as other toxic or malodorant gases or fumes:

1. The imposition of regulations prohibiting automobiles, in passing through a crowd or in the vicinity of other vehicles, animals, or pedestrians, from emitting fumes or odors of a nature to inconvenience the public, impede traffic, or, referring more especially to smoke, frighten animals; special supervision of garages and of public vehicles for the protection of users from the products of combustion. This regulation would also forbid the use of tetraethyl lead and its derivatives and would fix the responsibility on the owner of the automobile or the garage, and finally on the manufacturer, for death by intoxication.

2. The personal action of manufacturers and drivers, which could be greatly facilitated through collaboration with syndical chambers of the automobile industry.

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45 Safety in Mines Research Board, Eighth Annual Report: Colliery Guardian, vol. 141, No. 3632, August 8, 1930, p. 481.

46 Kohn-Abrest, and Loiret, Rapport Presenté au Nom de la Commission de la Préfecture de Police pour la Suppression des Fumees et des Gaz d'Echappement Nocifs des Automobiles: Ann. d'hygiene publique, industrielle, et sociale, vol. 7, New Series, July, 1930, pp. 373-410.

Germany.- An investigation[47] was made of the toxicology and hygiene of motor traffic, because of the increasing number of accidents due to inhalation of exhaust gases from motor vehicles. The investigation included chemical analyses of exhaust gases, toxicological experiments, and practical tests as to the amount of carbon monoxide, carbon dioxide, and methane, etc., present in over 100 street blocks, tunnels, and in large garages.

The general conclusion reached - and here the authors are in striking agreement with the decision arrived at in the Final Report of the English Ethyl Petrol Committee - is that carbon monoxide is the only gas with acute action given off by water vehicles. In consequence, however, of the extraordinary rapidity of dilution, the CO concentration in the atmosphere at breathing level rarely exceeds 2 parts per 10,000, so that risk to the general public is to all intents and purposes absent. In 95 of the 101 tests the proportion of CO was under 1.5 parts per 10,000. The highest was 2.7 which, it is said, closely corresponds with observatons made by investigators in the United States. The highest proportion found by Major R. A. Hepple for the Ethyl Petrol Committee was 1.7 parts per 10,000 (at Trafalgar Square), but when the sample was taken the bottle was placed quite close to the exhaust.

As inhalation of a mixture of 2 parts CO per 10,000 parts air if continued for hours together must lead to some saturation of the blood by CO and symptoms of headache and palpitation, precautions are called for in the case of police on point duty in places where "hold-ups" are frequent, and in garages and repair workshops.

South Africa.- The ventilation of the mines of the Witwatersrand[48] has four main objects in view: the supply of air chemically pure, the mitigation and removal of the danger from siliceous dust, the supply of cool air to the deep mines, and the removal of the dust and fumes produced by blasting.

In order to meet these requirements certain regulations, promulgated under the Mines and Works Act of the Union (Handbook of the Mines and Works Act of the Union, Hortors, Ltd., Johannesburg), are applied. As regards the chemical purity of the air, carbon dioxide must not exceed 0.2 per cent, carbon monoxide 0.01 per cent, and there must be no detectable trace of oxides of nitrogen. At least 30 cubic feet of air per minute must be supplied to every person employed underground.

United States.- A review of most of the important reports of studies made on carbon monoxide, since the existence of this substance has been known, was published by the U. S. Public Health Service.[49] This summarizes the information available on the occurrence, symptoms, and diagnosis of carbon monoxide poisoning, percentages of the gas dangerous to breathe, with symptoms that occur at various percentages of blood saturation, and the pathology, prevention, and treatment of poisoning by this gas.

47. Keeser, E., Froboese, V., Turnau, R., Gross, E., Kuss, E., Ritter, G., and Wilke, W., Toxikologie und Hygiene des Kraftfahrwesens (Auspuffgase und Benzine) (Toxicology and Hygiene of Motor Traffic.): Schriften Gewerbehyg., 1930, new ser., No. 29, pp. viii and 106. Quoted from Bull. Hygiene, vol. 5, No. 2, November, 1930, p. 898.
48 Rees, J. P., Mine Ventilation on the Rand: South African Min. and Eng. Jour., April, 1930, page 65.
49 Sayers, R. R. and Davenport, Sara J., Review of Carbon Monoxide Poisoning: Public Health Bull. 195, 1930, 97 pp.

A study was made by the Bureau of Industrial Hygiene, New York State Department of Labor[50] on the effect of chemically pure carbon monoxide, illuminating gas, and automobile exhaust gas upon the fragility of the red blood cells. When normal blood was exposed to pure carbon monoxide gas under laboratory conditions, there was no increase in the fragility of the red blood cells.

Normal blood exposed to both illuminating gas and automobile exhaust gas, under the same laboratory conditions, showed a tendency to a somewhat increased hemolysis of the red cells. The increase in hemolysis due to automobile exhaust gas is slightly less than that produced by illuminating gas.

The hemolyzing effect of these gases would seem therefore to be due to the presence of other toxic constituents rather than to their carbon monoxide content. The hemolysis produced by the several treatments outlined was not due to a change in the hydrogen ion concentration of the blood.

Carbon monoxide poisoning has been declared compensable when incurred during employment in a garage.[51]

According to Commander Brown,[52] of the Medical Corps of the U. S. Army, the disasters to the S-51 and S-4 within recent years have aroused renewed interest in the subject of methods of rescue for submarines. Among the various measures for rescue, a vital consideration is provision for the regeneration of the air in order that the crew may survive while rescue operations are being conducted. Commander Brown states the problem as follows:

> Soda lime is supplied to submarines to prevent dangerous accumulation of carbon dioxide, and oxygen is provided in the form of the compressed gas. The limitations of weight and space in submarines are such that these supplies are necessarily very restricted. As a result of careful investigation, information is now available as to the maximum absorptive efficiency of soda lime for CO_2 under the usual ventilation conditions; and the maximum time that the oxygen will last from the standpoint of survival is known for all practical purposes.
>
> There are, however, certain questions which will arise in naval circles. One of these may be formulated as follows: To what extent, if any, will the maintenance of the normal oxygen concentration minimize the effects of high carbon dioxide on men resulting from rebreathing air in submarines? Would life be materially prolonged if oxygen deficiency were prevented while carbon dioxide was accumulating to a dangerous percentage?

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50 Mayers, M. R., Rivkin, H., and Krasnow, F., The Effect of Chemically Pure Carbon Monoxide, Illuminating Gas, and Automobile Exhaust Gas Upon the Fragility of the Red Blood Cells: Jour. Ind. Hygiene, vol. 12, No. 8, October, 1930, pp. 300.

51 Jackson v. Euclid-Pine Inv. Co. (Mo.), 22 S. W. (2d) 849. Workmen's Compensation Acts: Carbon Monoxide Poisoning of Garage Employee Compensable. Jour. Am. Med. Assoc., vol. 95, No. 7, August 16, 1930, p. 554.

52 Brown, E. W., The Value of High Oxygen in Preventing the Physiological Effects of Noxious Concentrations of Carbon Dioxide: U. S. Naval Bull., vol. 28, No. 3, July, 1930, pp. 523-553.

Another important question is the effect of high carbon dioxide with or without oxygen deficiency on the physical and mental efficiency of men in submarines. The efficiency curve will fall, of course, when the atmosphere is vitiated by a high concentration of CO_2. It should, however, be determined, if possible, what concentraion would induce effects leading to such a loss of efficiency that the boat could not be properly operated by the personnel. It is also possible that mental efficiency may be more seriously affected than physical stamina and may occur earlier. There is apparently no record in the literature of the effect of high CO_2 on mental efficiency, although some work has been carried out in reference to physical efficiency.

An investigation was carried out to determine (1) the noxious effects of carbon dioxide with oxygen supplied to prevent deficiency and with the oxygen allowed to fall as the carbon dioxide accumulated, and (2) the influence of high carbon dioxide on physical and mental efficiency of the personnel. The investigation showed that such subjective effects as panting, dyspnea, headache, nausea, and chilliness and fatigue were somewhat less in the high oxygen group. The differences, however, were not striking, the more decided being reduction of dyspnea and less fatigue during and after the tests. The following statement is made of the effects of high carbon dioxide:

> It is felt that the personnel could carry on their usual submarine duties for a protracted period, if carbon dioxide did not exceed 5 per cent. Even at approximately 6 per cent of carbon dioxide the men could probably still carry on for a short time, the efficiency curve falling rather rapidly between 5.5 and 6 per cent. It is believed that the majority would be completely incapacitated above 6 per cent of carbon dioxide, which is regarded as the critical point. The supply of oxygen would improve physical and mental efficiency between 5 and 6 per cent of carbon dioxide but would not prolong it beyond the latter figure.

A consideration of the hazards to health and safety is an important feature in the innovation of chemicals which may have rather wide use under conditions where persons are exposed to air containing their vapors. Frequently, however, the information necessary for a basis of evaluation of the hazards is lacking, due mainly to the materials being relatively new products, or at least new to the particular field or conditions of use. In view of this, there is a continual need of research and investigations to supply the information, especially at the present time, when there is considerable activity in the development of new organic compounds of domestic and industrial importance. Fortunately, along with the activity in development, progressive chemical industries have realized the importance of the health aspects in the manufacture, marketing, and utilization of their products, and many have initiated and supported research to that end.

The U. S. Bureau of Mines,[53] in cooperation with several of the large chemical companies, has carried out an investigation of the physiological response of guinea pigs to ethylene dichloride, ethyl benzene, "cellosolve," ethylene oxide, vinyl chloride, and dioxan. From the standpoint of the health hazard, ethylene dichloride and "cellosolve" are about as harmful as gasoline, benzene, carbon tetrachloride and chloroform; ethyl benzene and vinyl chloride are slightly less harmful than these gases, and ethylene oxide is less harmful than hydrogen chloride and sulphur dioxide, but a great deal more harmful than carbon tetrachloride and chloroform. The hazard to health from breathing air contaminated with dioxan is slight. Health hazards from dioxan, ethyl benzene, ethylene oxide, ethylene dichloride and cellosolve are mitigated by the warning response manifested as eye and nose irritation. Ethylene dichloride has a distinct odor and cellosolve a disagreeable one. Vinyl chloride does not possess adequate warning properties, but gives warning by producing dizziness and disorientation in advance of harm. In the use of new substances of this kind it is always recommended that in so far as possible exposure should be reduced to a minimum, and unavoidably exposed workmen should be regularly given complete physical examination. Nearly all organic vapors are toxic and present potential health hazards, and much remains to be learned about the effects of repeated exposure to relatively small amounts.

In connection with the study[54] made on the toxic effects of materials used for mechanical refrigeration, reported last year, attention was called to the fact that the hazard to health from contamination of air by a noxious gas depends not only on the potential harmful response attending exposure, but also on the warning properties, which in effect may be termed the "accompanying warning response." This warning may be manifested as an odor, taste, irritation of the eyes, nose, or throat, or perhaps headache, vertigo, or nausea. These manifestations may be painful and even slightly harmful, but to a much lesser degree than the primary injury of exposure. The warning response mitigates hazards directly in proportion to the degree of intolerability which accompanies injurious exposure, and accordingly it's absence augments hazards. This accounts for the fact that the actual health hazards from a nonodorous or nonirritating gas or vapor of comparatively low toxicity sometimes equal or exceed the hazard from a considerably more toxic substance, but one which possesses a marked odor or produces eye, nose, or throat irritation in advance of harmful exposure.

When a substance lacks the properties for giving warning, it is often possible to impart them by the addition of a small amount of another substance that has an exceedingly high warning intensity.

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53 Acute Response of Guinea Pigs to Vapors of Some New Commercial Organic Compounds:
- I. Ethylene Dichloride
  Reprint No. 1349 from the Public Health Reports, 1930, 16 pp.
- II. Ethyl Benzene
  Reprint No. 1379 from the Public Health Reports, 1930, 10 pp.
- III. "Cellosolve" (Mono-Ethyl Ether of Ethylene Glycol)
  Reprint No. 1389 from the Public Health Reports, 1930, 8 pp.
- IV. Ethylene Oxide
  Reprint No. 1401 from the Public Health Reports, 1930, 12 pp.
- V. Vinyl Chloride
  Reprint No. 1405 from the Public Health Reports, 1930, 9 pp.
- VI. Dioxan
  Reprint No. 1407 from the Public Health Reports, 1930, 10 pp.

54 Sayers, R.R., Yant, W.P., Thomas, B.G.H., and Berger, L.B., Physiological Response Attending Exposure to Vapors of Methyl Bromide, Methyl Chloride, Ethyl Bromide, and Ethyl Chloride. Public Health Bull. 185, 1929, 56 pp.

The manufacturers of methyl chloride and a manufacturer of refrigerating devices investigated the properties and possibilities of use of an exceedingly large number[55] of substances and selected acrolein as the most promising. These companies, after completing their investigation, asked the Bureau of Mines to make an independent study of the efficacy and suitability of acrolein. Two reports[56] were made, one for unit systems and one for multiple systems.

The studies made by the Bureau of Mines showed that exposure to 1 part of acrolein per million parts of air produces detectable eye and nose irritation in two to three minutes, moderate eye irritation with lacrimation in four minutes, and is painful and practically intolerable in five minutes. Concentrations of 5.5 parts per million of air cause painful eye and nose irritation in 20 seconds and are practically intolerable in one minute. One part per million is thought to be adequate for giving warning. When charged with methyl chloride containing 0.6 to 1.0 per cent of acrolein by volume, the amount of acrolein in the various parts of the commercial units tested was enough to give warning of leakage of liquid from the evaporator or float chamber when the concentration of methyl chloride in the air was 0.0066 to 0.01 per cent by volume, and 0.066 to 0.10 methyl chloride if the leakage was vapor from the remainder of the system. Animal experiments have indicated that repeated daily exposure of several hours to 0.005 to 0.01 per cent does not cause apparent harm and that 0.05 to 0.10 does not cause apparent harm after a single exposure of several hours.

In connection with the above studies on atmospheric contamination, attention was also given to possible contamination of food and poisoning by ingestion. Following is a summary[57] of this investigation:

The possibility of poisoning by ingestion of methyl-chloride contaminated food was studied by exposing dogs.

> No apparent signs of poisoning were caused by the average daily ingestion on four consecutive days of 500 grams of ground raw beef or 200 c. c. of milk that had been exposed 15 to 75 hours to 100 per cent methyl-chloride vapor at 35°F.

A report was also issued by the Bureau of Mines[58] on the use of ethyl mercaptan to detect leaks in natural-gas distribution systems. The bureau had been desirous of observing the practical use of ethyl mercaptan for detecting leakage of natural gas, particularly to determine whether the odor would permeate the ground and give indication of leaks in underground pipes.

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55 Roessler & Hasslacher Chemical Co., Warning Agents for Methyl Chloride in Refrigeration Systems: 10 East 40th St., New York City, 1930, 31 pp.

56 Yant, W. P., Schrenk, H. H., Patty, F. A., and Sayers, R. R., Acrolein as a Warning Agent for Detecting Leakage of Methyl Chloride from Refrigerators; Rept. of Investigations 3027, Bureau of Mines, 1930, 11 pp.
Schrenk, H. H., Patty, F. A., and Yant, W. P., Acrolein as a Warning Agent for Detecting Leakage of Methyl Chloride from a Multiple Refrigeration System: Rept. of Investigations 3031, Bureau of Mines, 1930, 7 pp.

57 Yant, W. P., Shoaf, H. W., and Chornyak, J., Observations on the Possibility of Methyl Chloride Poisoning by Ingestion with Food and Water: Reprint No. 1371 from the Public Health Reports, 1930, 8 pp.

58 Sayers, R. R., Fieldner, A. C., Yant, W. P., Leitch, R. D., and Pearce, S. J., Use of Ethyl Mercaptan to Detect Leaks in Natural-Gas Distribution Systems: Rept. of Investigations 3007, Bureau of Mines, 1930, 13 pp.

Opportunity for making these observations was recently given by E. A. Munyan, manager, gas department, Union Gas & Electric Co., who requested the bureau's cooperation in making leakage surveys of the company's distributing systems at Franklin and Middletown, Ohio.

In these leak surveys the use of ethyl mercaptan was entirely successful. It appeared to be far more efficient than ordinary inspection methods and entailed only a fraction of their cost. A discussion is given of the chemical and physiological properties of ethyl mercaptan, the procedure for conducting ethyl mercaptan surveys, the importance of favorable season and weather conditions, and the concentrations found to be effective. The odorizing of natural gas with ethyl mercaptan was found to be a practical means for detecting leakage and a much cheaper means than usual inspection methods. Concentrations of 7.7 to 9.3 pounds of ethyl mercaptan per million cubic feet of gas were found very effective in indicating house leaks. It is thought that half that amount or even less would be ample to indicate leaks of significant magnitude. Concentations of 31.0 to 46.5 pounds of ethyl mercaptan per million cubic feet of gas were found effective in indicating underground leaks in mains and service lines. The use of ethyl mercaptan caused no complaints from customers unless leaks were present.

SUMMARY OF RECENT LITERATURE ON ABNORMAL TEMPERATURES AND HUMIDITIES

The subject of the control of atmospheric conditions indoors, especially the temperature and humidity, is becoming of increasing importance in connection with the health and efficiency as well as with the comfort of the individual. An air-conditioning engineer[59] has expressed the following ideas concerning the developments that lie ahead in the "manufacture of weather":

> We are fast approaching the time when the average suburbanite will rise in the morning refreshed from sleeping in an automatically controlled and carefully regulated atmosphere. He will breakfast in a house whose temperature and humidity are conducive to good nature and sound nerves. He will tavel to the city in an air-conditioned railroad train from which objectionable odors and overheated dry air are eliminated. Upon reaching his office he will enter the invigorating atmosphere of a type of weather manufactured to suit him and lunch in comfort under the same condition.

59 Carrier, W. H., The Future of Air Conditioning: Quoted by Floyd W. Parsons in Facts and Fancies, Gas Age-Rec., November 8, 1930, p. 770.

Japan.- The significance of sweating in man has been studied by Yas Kuno and his collaborators for eight years at the Physiological Laboratory of Manchuria Medical College.[60] He mentions the current view, held for many decades, that the regulation of the temperature of the body is almost the sole object of sweating, which he has long doubted since sweating may be provoked not only by an increase of temperature of the surroundings but also by emotion and mental stress. The reason for the increase in sweating due to environment is clear, but why emotion or mental stress should also cause sweating he thought should be investigated. He found that sweating caused by heating inevitably spreads over the entire body, but the palms of the hands and the soles of the feet do not conform to this general rule. However, any intense sensory stimulation seems able to provoke sweating in these parts. The most adequate cause was found to be mental stress. Mental arithmetic caused sweating on the palms of the hands and the soles of the feet. Kuno, therefore, classifies the human sweat glands in two groups with respect to causation as well as to distribution on the body surface:

(1) The sweat glands present on the palms and the soles have the characteristic of permanent secretion and react with an increased secretion to mental or sensory stimulation but not to rise of temperature in the environment.

(2) The sweat glands distributed all over the remaining parts of the body surface show little or none of this permanent secretion or "insensible perspiration." They are not responsive to moderate mental stimulation, but profuse sweating occurs from them when the temperature of the environment rises. One phenomenon is, however, common to these two different sets of sweat glands - i.e., every sweating, whenever it occurs, appears universally over its own respective district and is never confined to any single part in its area. For brevity I will designate the sweating from the former set of sweat glands as "mental or psychical sweating," and that from the latter as "thermal sweating."

He concludes that the sweat glands of human beings have three important functions: (1) The regulation of the body temperature; (2) the facilitation of physical work; and (3) the protection of the skin.

United States.- Ventilation is not air conditioning, according to Stangle and Kingsbury,[61] although a space equipped with an air conditioning system and maintaining the desired standard synthetic atmosphere is adequately ventilated. They distinguish between ventilation and air conditioning as follows:

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60 Kuno, Yas, The Significance of Sweating in Man: Translated and published in The Lancet, vol. 1, No. 17, April 26, 1930, pp. 912-915.

61 Stangle, W. H., and Kingsbury, H. W., Air Conditioning: IV. Ventilation-General Discussion: Heating and Ventilating, July, 1930, p. 78, August, 1930, p. 75.

Ventilation is the supply, circulation and distribution of fresh air and the accumulation, circulation and removal of vitiated air. The removal of foreign matter such as dust and dirt by means of filters may be considered as a part of ventilation but borders on conditioning of air since we are doing something to the air that changes its characteristics.

Let us, therefore, say that when we do anything to air to change its characteristics, we condition the air. Such treatment of air may be filtration, heating, washing, cooling, etc. Ventilation on the other hand, as we have stated, is the movement of, and the handling of, air in its natural state. That is why the modern phrase "synthetic atmosphere" has been coined, since it is the man-made product resulting from conditioning air.

If a space has a comfortable temperature, is free from odors, and has a gentle but definite air movement with no drafts it is well ventilated.

Supplying a certain quantity of air whether it is 2 c.f.m. per sq. ft. of floor area or 30 cu. ft. per person, does not mean that the space is well ventilated. It must be comfortable.

Comfort is most important. It is Nature's way of telling us that our immediate surroundings are satisfactory for our health. If there are elements present which dull our senses, this is not true. Overheating does not cause the same degree of discomfort as underheating or drafts and is not as dangerous to the health.

The American Society of Heating and Ventilating Engineers and the Bureau of Mines have made an interesting study of comfort with varying temperature, humidity, and air movement conditions. A new standard is used in defining a comfortable temperature. It is called the Effective Temperature. We base our calculations on an effective temperature of 67° which is equivalent to 70° dry bulb and 60° wet bulb, a too high humidity. A temperature of 72° dry bulb and 57° wet bulb, which has the same effect, is to be preferred.

In summer the temperature inside should not be lower than 10° below that outside. Comfort in summer is comparative and is varied by the amount of clothing worn and other conditions. A maximum temperature of 85° with a relative humidity of 50% is commonly assumed. This summer comfort zone will be discussed in further detail.

The effects of too dry air and the suggested air conditioning for persons suffering with respiratory diseases are described by Barnum[62] as follows:

> Dr. Herman N. Bundesen, for many years Commissioner of Health in the City of Chicago, is perhaps one of the best known of our Public Health officials who has made a real study of this subject and has written quite at length on it. He says: "Proper moisture, or what is known as humidity, is not merely desirable, but is essential to health and comfort. Humidity and temperature are in close relation and both must always be considered in connection with ventilation."
>
> Respiratory diseases are the most common of our human ailments affected by temperature and humidity. Contrary to the belief of most people, "the breathing of cold air affects the mucous membranes of the nose in such a way as to encourage the invasion of disease germs," according to Dr. J. A. Meyers, Ph.D., M.D., professor of preventive medicine and chief of chest clinic of the University of Minnesota, Minneapolis. To continue quoting Doctor Meyers: "In the treatment of the acute respiratory diseases such as coryza and bronchitis, cold air has an unfavorable effect. How often we see a patient who complains of frequent prolonged colds terminating in bronchitis which may become sub-acute. The ordinary methods of treatment have proved of no avail but, upon inquiry, we find that he is sleeping out of doors or in an extremely cold room, and will continue to have these colds until he can be convinced that he should breathe reasonably warm air with the proper humidity. With our new knowledge of the effects of the air upon the body, we are fast passing the day when pneumonia is treated in a very cold room. Better results are obtained when the air in the room is kept in the neighborhood of 68° with a humidity of 40% to 50% than when the windows and doors are thrown wide open and the patient is subjected to great exposure."
>
> Doctor Meyers holds to the same theory regarding the treatment of tuberculosis, namely that the patient is better off at a temperature of 68° F. and a relative humidity of about 40% than in the extreme cold. He speaks of the "tremendous amount of energy required on the part of the body to keep warm" and believes that this is an unnecessary tax upon the patent.

In a report of the Committee on the Atmosphere and Man of the Natural Research Council, Huntington[63] gives the following results of a study on the connection between weather and health in New York City:

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62 Barnum, M. C., Effects of Too Dry Air: Heating and Ventilating, June, 1930, p. 83.

63 Huntington, Ellsworth, Weather and Health: Bull. Nat. Research Council, No. 75, 1930, 161 pp.

The results here set forth agree in general with those of other investigators, but add important new features. They confirm the idea that man, like other animals, is subject to a distinct optimum of temperature. When all ages and all causes of death are combined, this optimum, as derived from the present investigation, falls at a temperature of 63° F. for day and night together. This is close to the average among the various determinations of the optimum, but one of the new things brought out by the present report is that the optimum varies greatly for different ages of life, and with respect to resistance to different diseases. Deaths from pneumonia (and influenza) decline with almost perfect regularity as the temperature rises; those of children under five years of age decline similarly but far less rapidly until the average temperature reaches 55°, but increase portentously with every rise of temperature above 60°.

Variable weather at all seasons, in New York City at least, is accompanied by a lower death rate than is uniform weather. Extremely variable weather, to be sure, is not so favorable as that which is intermediate, but it is distinctly better than that in which day after day is of the same temperature.

When temperature, humidity and variability are all combined it appears that the ideal day is one on which the average temperature is about 65° and the relative humidity nearly 90 per cent. The preceding ten days or so should have been characterized by fairly strong changes of temperature averaging 4°. These should culminate in a drop of 10 or 12°. Of course such days can not occur continually, for a further drop will carry the temperature to an unfavorable level. A rise will do likewise and will itself be unfavorable. No change of temperature will also work harm. Yet in spite of all this it is possible for days which approximate the optimum to occur frequently.

As a part of a study of respiratory illness among steel workers (pneumonia occurs with almost twice the frequency among iron and steel workers as among the employees of a group of miscellaneous industries) the U. S. Public Health Service compared the skin temperatures of workers exposed to radiant energy with the skin temperatures of workers not so exposed. Preliminary analysis of the cases occurring over a four-year period had indicated that the incidence of pneumonia was high among workers exposed to inclement weather and among those subjected to wide variation in temperature, especially extreme heat followed by exposure to much lower temperatures. The main purpose of the preliminary report[64] has been to describe the instruments employed in the field to determine radiant energy and skin temperature. Incidentally, however, a general picture of the results obtained has been given as an indication or sample of the relationship between skin temperature, radiant energy, and the atmospheric conditions among steel workers. The final comparisons have been left to a later monograph, which will show what correlations may appear between conditions found and the sickness rates, primarily pneumonia. The following points have come out rather clearly in this preliminary analysis:

64 Bloomfield, J.J., Ives, J.E., and Britten, R.H.: Effect of Radiant Energy on the Skin Temperatures of a Group of Steel Workers; Reprint No. 1370, Pub. Health Repts., 1930, 13 pp.

1. Intense sources of radiant energy had a pronounced effect on the skin temperatures of workers exposed to them; the forehead and cheeks showed the greatest increase.

2. Great differences in the skin temperatures of different parts of the body, for a single individual, were found in workers exposed to radiant energy.

3. Even under relatively cold atmospheric conditions, not far above the freezing point, high skin temperatures were encountered in workers exposed to radiant energy.

4. For workers not exposed to radiant energy there was a definite relation between atmospheric conditions and skin temperatures, both for arduous and for moderate work; the skin temperatures increasing with increase of effective temperature.

Tests made by Ward[65] on the measurement of skin temperature in relation to the sensation of comfort, indicate that the temperature of the skin as measured over the forehead and carotid may be used as an index of the degree of comfort of the individual. While there are undoubted individual variations, the measurements show in general that an optimum sense of comfort is experienced when the skin temperature at these points lies between 33 and 34° C. An uncomfortable sensation of warmth is reached when the skin temperature rises as high as 35 to 36°C. On the other hand, uncomfortable sensations of cold are experienced when the skin temperature falls as low as 31 or 32° C. Fewer data were obtained for this lower limit, which should be studied with more care. On the basis of these results, one might place the extremes of the comfort zone for the skin temperature of the forehead or carotid as lying between 31.5 and 35.5°C., a range of 4° C. with the optimum at 33.5° C. Beyond these limits distinct discomfort is felt. These results apply to winter conditions in a temperate climate. It would be interesting to determine whether under summer conditions or in tropical climates these limits would show an alteration in consequence of acclimatization.

65 Ward, Emma F., The Measurement of Skin Temperature in Its Relation to Comfort: Am. Jour. Hygiene, vol. 12, No. 1, July, 1930, pp. 130-154.

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DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MINING METHODS AND PRACTICES AT THE UNITED VERDE COPPER MINE, JEROME, ARIZONA

BY

T. W. QUAYLE

I.C.6440,
February, 1931

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AND PRACTICES AT THE UNITED VERDE COPPER MINE, JEROME, ARIZ.[1]

By T. W. Quayle[2]

INTRODUCTION

This paper describing mining practices at the United Verde copper mine, Jerome, Ariz., is one of a series being prepared by the Bureau of Mines on mining practices, methods, and costs in the various mining districts of the United States.

At present mining operations are being conducted both underground and by electric shovels in an open pit. The open-pit work has been described by E. M. J. Alenius in a previous publication of the Bureau of Mines;[3] the present paper will be confined to a description of the underground mine. The underground methods comprise horizontal cut-and-fill, shrinkage with delayed filling, horizontal square-set, inclined square-set, and top slice. The normal daily production of the underground mine is 3,000 tons of copper ore per day, requiring the employment of 900 men underground.

ACKNOWLEDGMENTS

The author wishes to acknowledge the assistance given by W. W. Lynch, formerly general mine superintendent, Carl E. Mills, assistant mine superintendent, M. G. Hansen, chief mine geologist, W. P. Goss, chief bonus engineer, O. A. Glaeser, ventilation engineer, J. F. Cowley, planning engineer, and others of the United Verde staff in the preparation of the subject matter and illustrations in this paper.

HISTORY

The United Verde mine is sutuated at Jerome, Ariz., on the eastern slope of the Black Hills, at an elevation of approximately 5,500 feet. The town of Jerome is served by the Verde Tunnel & Smelter Railroad, a subsidiary of the United Verde Copper Co., which joins a branch of the Santa Fe Railroad at Clarkdale, 4 miles distant in an air line from Jerome. At Clarkdale is the United Verde smelter.

The original claims covering the United Verde outcrop were located in 1876 by M. A. Ruffner. The outcrop contained oxides of copper and a considerable amount of gold and silver. However, as the terminal of the Santa Fe at that time was in Kansas and the nearest smelter was at San Francisco, the outlook for profit was dark. The United Verde Copper Co. was organized in 1882 and purchased the original claims from Ruffner for $45,000. James A.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used. "Reprinted from U. S. Bureau of Mines Information Circular 6440."

2 - One of the consulting engineers, U. S. Bureau of Mines, and mine superintendent, United Verde Copper Co.

3 - Alenius, E. M. J., Methods and Costs of Stripping and Mining at the United Verde Open Pit Mine, Jerome, Ariz.: Information Circular 6248, U. S. Bureau of Mines, February, 1930, 33 pp.

McDonald and Eugene Jerome of New York were made, president and secretary, respectively. A small reduction plant was installed and shipments of high-grade gold-silver-copper matte began. Financial and operating difficulties, however, caused the operation of the mine and plant to be spasmodic for several years.

Senator W. A. Clark of Montana made his first visit to the mine in 1888, having been impressed by an exhibit of its ores at the World Exposition at New Orleans in 1885. After personally examining the property he secured an option and took control in 1889. Under his guidance developments were rapid and the United Verde eventually became one of the world's leading copper mines.

Senator Clark remained president of the company until his death in 1925. He was succeeded by his son, Charles W. Clark, who now heads the company.

GEOLOGY AND ORE DEPOSITS

The ore deposits of the United Verde mine are of the massive sulphide, schist-replacement type. A structural anticline formed by the intrusion of diorite between bedding planes of steeply-dipping and severely-folded greenstones and porphyry localized the deposition of sulphide. The mineralized zone is of great horizontal extent, varying from 200,000 to 400,000 square feet on various levels, and has a proved depth of 4,000 feet. The irregular inter-fingering of the sulphide with the schists and porphyry, typical of replacement types of ore deposits, forms an irregular and indefinite footwall boundary. The chief primary copper mineral is chalcopyrite; the gangue consists principally of pyrite, quartz, and chlorite.

General Geology

The geologic record begins in early pre-Cambrian times, when a greenstone complex consisting of tuffs, flows, and fragmentals was formed. Younger than the greenstones is a series of siliceous bedded sediments. After the deposition of the latter the formations were folded and tilted steeply to the northwest. Following this the area was intruded by rhyolitic quartz porphyry, which was shortly thereafter rendered schistose by further deformation. Masses of augite diorite then intruded the quartz porphyry and greenstones. The sulphide ore deposits were formed after the intrusion of the diorite. After the formation of the large schist replacement ore bodies, but before the mineralization was entirely completed, a series of small andesite dikes, locally called "water courses," cut the ore masses and other formations. These dikes have an east-west trend which is nearly at right angles to the schistosity of the quartz porphyry. They vary in thickness from a few inches to 50 feet, with an average of 2-1/2 feet.

Continued deformation following the intrusion of the dikes brecciated and displaced all of the older formations. Normal faulting on a major scale caused a vertical displacement of about 2,300 feet on the Verde fault. This fault strikes north-northwest south-southeast. It cut the chimney of sulphide into two segments. A long period of erosion followed, during which the country was base-leveled. It was later covered by the Cambrian seas, which deposited the Tapeats sandstone 0 to 100 feet in thickness.

Overlying the basal sandstones are 300 to 500 feet of Devonian limestone, 300 to 500 feet of Mississippian limestone, and 0 to 500 feet of red sandstone of Permian age. Each of the four periods of deposition was preceded and followed by periods of uplift and erosion.

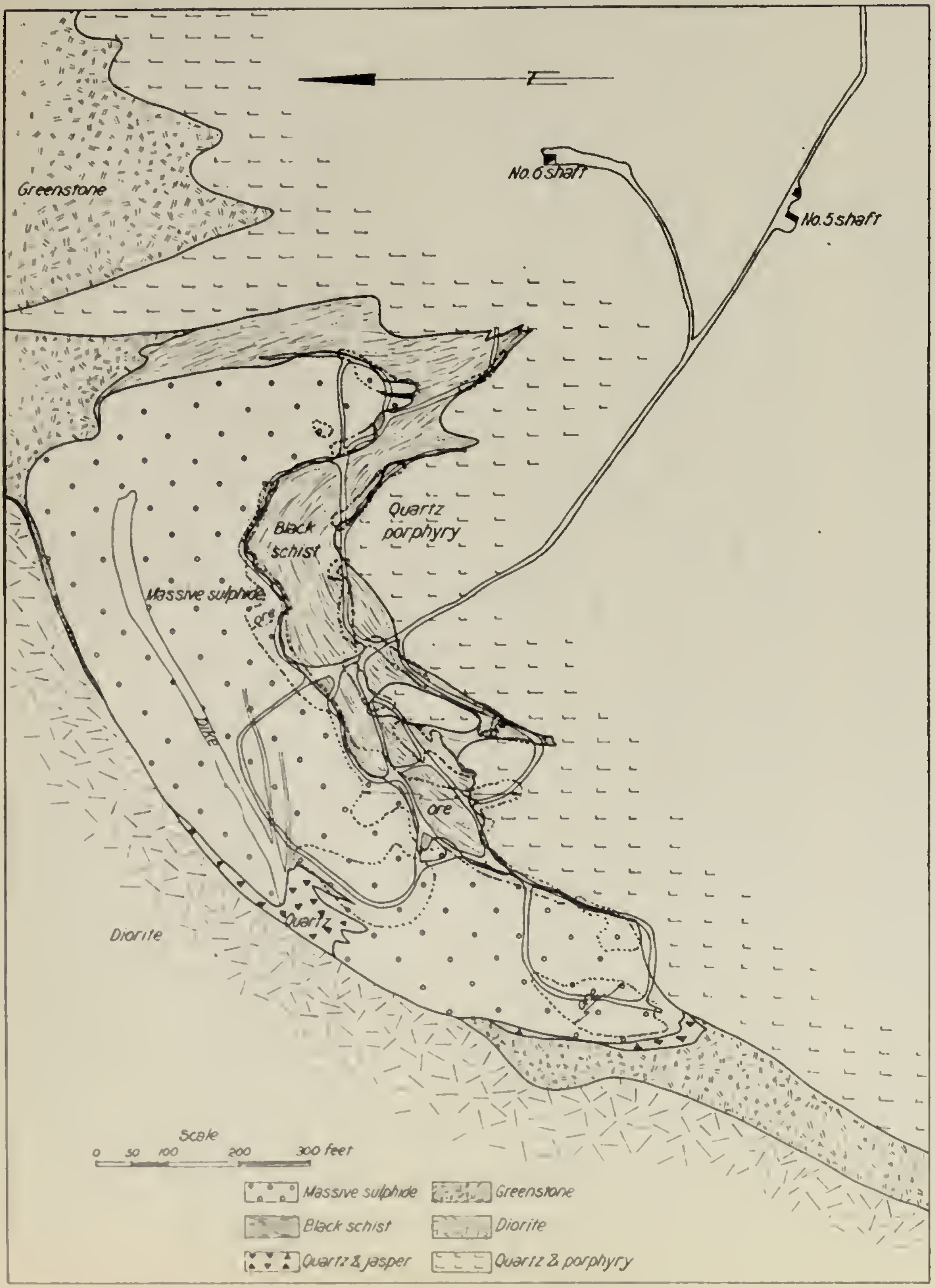


Figure 1.-Geologic map of typical level

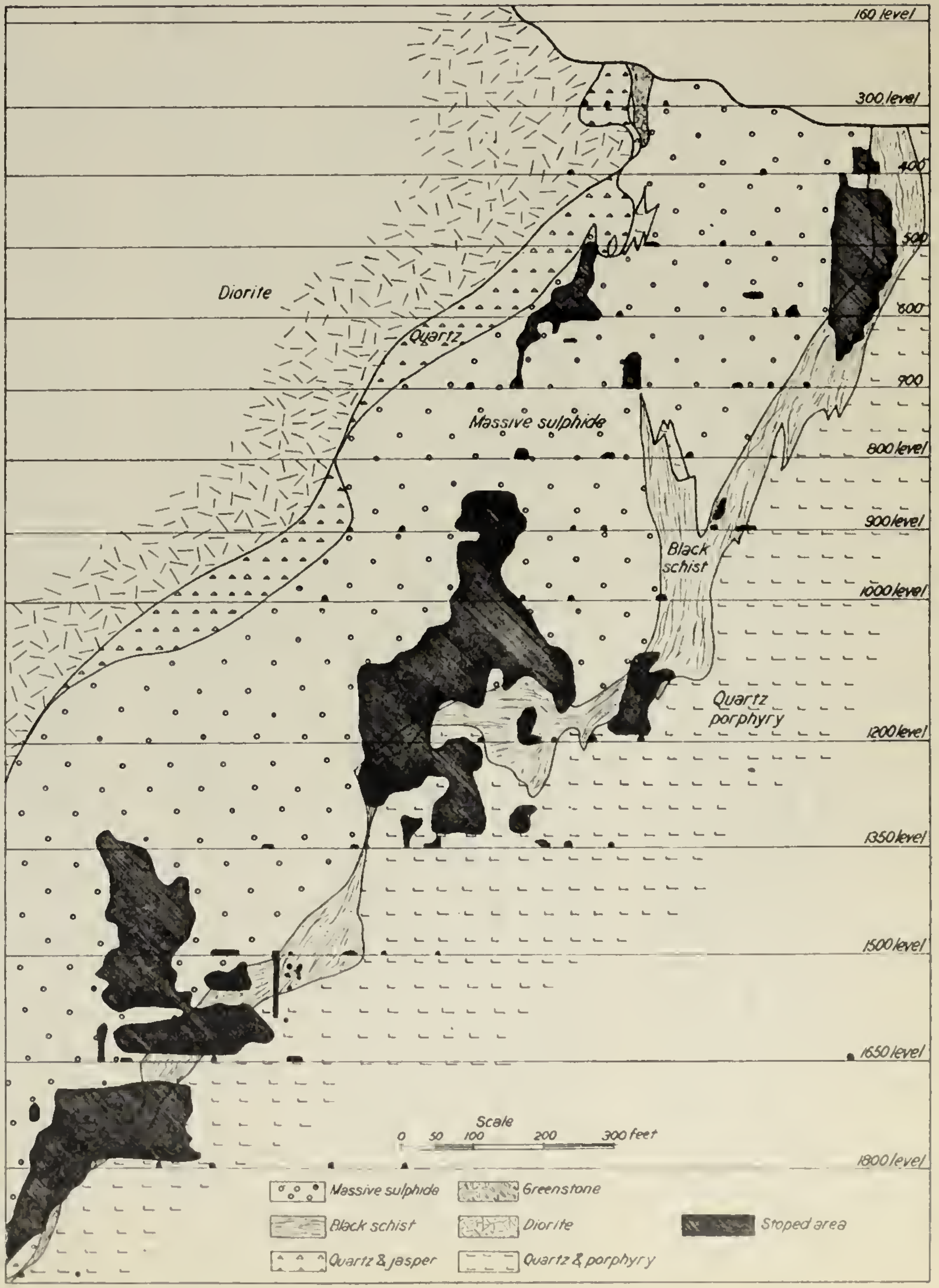


Figure 2.-Geologic vertical section

Tertiary basaltic lavas are the next recorded formations. These came chiefly from the San Francisco Mountains volcanic field, augmented somewhat by feeders in the form of dikes which are occasionally found in the district. Where the lavas flowed across the Tertiary Verde Valley below Camp Verde, a dam was formed which resulted in a lake at least 35 miles long and 6 or 8 miles wide. In this lake bed (Verde) formations were deposited to a thickness of at least 1,200 feet.

Following the outpourings of lava and more or less contemporaneous with the deposition of the lake bed formations came another period of normal faulting. Several breaks are apparent with a north-northwest and south-southeast trend, also a few attendant cross faults. The Verde fault moved again, causing an additional vertical displacement of about 1,580 feet. The uplifted scarp of the Verde fault was especially subject to erosion so that a strip of the pre-Cambrian was exposed. The erosion on the west side of this fault uncovered the ore bodies of the United Verde mine, whereas on the east side of the fault the pre-Cambrian formations are deeply buried.

Due to the rapid erosion of the steep fault scarp and the ore bodies, very little secondary enrichment exists. Erosion has followed enrichment so closely that the oxidized capping does not extend over 100 feet in depth, and secondary enrichment rarely 200 feet below the oxides.

The ore deposits are of the schist-replacement type. Where the massive sulphides adjoin schists there is a gradual transition from unreplaced to completely replaced schist. The contact is rarely a clean-cut one, and shows all the embayments and irregularities characteristic of limestone replacement deposits. Fingers, pendants, and small isolated patches of ore frequently exist. The ore bodies generally occur along the sulphide-schist contact, and to a less extent within the sulphide, schist, or porphyry areas (see figs. 1 and 2).

The sulphide varies in horizontal section from an elliptical to a roughly crescent-shaped mass with a length of about 1,100 feet along the strike and an average width of about 300 feet. It lies between the chloritic-schist footwall and the diorite. Toward the diorite the sulphide becomes hard and siliceous, with practically solid jasper at the diorite contact. The diorite is fresh and hard, with a fairly uniform texture and without much evidence of alteration. The black schists consist essentially of chlorite. The quartz porphyry as a rule is fresh and quite hard, though in places in zones of shearing it may become very schistose and grade into black schist. Occasionally, near the sulphide body where acid leaching has had its effect, the porphyry is altered, softened, and kaolinized.

Physical Characteristics of Ore and Enclosing Rocks

As explained under the heading "Geology," the main ore bodies occur along the contact of the massive sulphide and the schist, with smaller and less important ones on the schist-porphyry contact or entirely in any one of the three rocks. The physical characteristics of the three rocks are so entirely different that the location of an ore body with respect to these rocks determines not only the mining method, but also the size of the stope than can be worked.

The massive sulphide is generally very hard and dense, and stands well over large areas without timber support, except for an occasional bulkhead under a loose slab or where weakened by an andesite dike. This permits mining by cut-and-fill and shrinkage methods

The schist varies so much in hardness and strength in different parts of the mine that no definite mining method can be assigned to the ore bodies in this rock. At places it stands well enough to be mined by the cut-and-fill method, whereas at other places it requires mining by small square-set sections. Shrinkage is never used in the schist because of dilution from the walls in drawing the broken ore.

In hardness and strength the porphyry forms an intermediate class between the massive sulphides and the schist, and the ore bodies in it can ordinarily be mined by the cut-and-fill method, except in a few places where the porphyry tends to break down in large blocks, leaving the back of the stope so ragged that timbering is required.

There is no general change in the characteristics of any one of the three rocks from the surface to the 3000 level, the present bottom of the mine.

In the massive sulphide and porphyry ore bodies the values shade off from commercial ore to waste, and the stoping limits are determined by assay. This may be true of the schist ore bodies, but more often the change from commercial ore to barren waste is sudden.

METHODS OF PROSPECTING AND EXPLORATION

Ore bodies may be found within an extremely large area, and the so-called "main ore zone" consists of numerous irregular lenses of ore, large and small. Therefore, various methods of prospecting are used, depending upon the area to be prospected and its location geographically and geologically. Diamond drilling is the chief method, followed by drifts, crosscuts, and raises.

The preliminary work in new and undeveloped country is done by diamond drilling, generally using flat holes, to obtain geological data. Should mineralized areas be found, or should favorable structural conditions be indicated, the drilling is followed by drifts and crosscuts. The footage of diamond drilling devoted to this class of prospecting amounts to about 5,000 feet annually.

Within the "main ore zone" the geology and ore areas are projected downward to each new level with sufficient accuracy to permit planning the haulage drift and the "contact" drift. The latter is so named because it is driven on or near the sulphide-schist contact. On the most recent "lift" of new levels the "contact" drifts were driven on the schist side of the contact and on definite bearings, as against the old practice of actually following an irregular replacement contact. This drift is guided by means of occasional diamond drilling, as well as by constant geological observations. Upon completion of this drift, the foot and hanging walls are thoroughly drilled, delineating the ore areas accurately so that the foot and hanging-wall drifts may be driven and the method of mining, with its stope and pillar system, may be planned.

Even after stoping operations have commenced, it is frequently necessary to diamond drill from the stopes with flat or up holes to locate the bottom of hanging pendants of ore which do not reach the level. More often than not the pitch and dip of ore shoots are irregular and must be determined in advance of stoping operations in so far as possible, to permit proper chute spacing. Numerous slips occur, which but rarely limit the ore shoots. Therefore, if a stope is bounded on a side by a slip, it is necessary to prospect beyond the slip at regular floor intervals with either short stope drifts or drill holes until it is assured that a false wall does not exist. To meet the requirements of stope exploration

a light, compact diamond drill was designed and constructed, weighing about 500 pounds and capable of drilling holes up to 250 feet in depth.

Five diamond drills are operated underground, drilling holes varying from "ES" to "N" in size (7/8 to 2-inch cores). The annual diamond-drill footage is 25,000 to 30,000.

METHODS OF SAMPLING AND ESTIMATION OF TONNAGES AND VALUE

Sampling

Five samplers and a head sampler are employed underground. Faces are usually sampled either by chipping with a sample pick or, to a less extent, by cutting channels. The chips are taken in four lines across the face for the width of the sample, beginning at the back, each line being successively lower. The average size of a sample is about 4 pounds. Individual pieces of rock or ore are limited to 1 inch in diameter. A sample covers a width of 5 feet, this width being regularly adhered to both in drifts and stopes. Usually the height of the wall or face sampled is 7 feet. The average length of a stope round is 6 feet. In the stopes, samples are taken after each round, so that a sample represents a horizontal area of about 30 square feet or about 20 tons.

Sampling errors vary with the type of ore. In the massive sulphides, the chalcopyrite or other ore minerals are very uniformly distributed and the copper content is gradational from the center of the sulphide ore area or the schist contact to the margin of the stopes, where the ore becomes noncommercial. Consequently, in sulphide, the sampling error is very low--an average of 2 per cent above actual content.

In the schist and porphyry ore areas, the occurrence of the ore minerals is extremely erratic, and, though in large areas of schist and porphyry a gradation of copper content may be noted, more often than not it is very rapid and sometimes abrupt. Oftentimes, too, in vertical extent ore and waste may occur on alternate floors for several floors in succession. Naturally then, the sampling error is high and can be reduced only by taking more samples. Possibly, as the ore minerals stand out in the matrix of porphyry and black schist, the personal element enters into the sampling of this class of ore. This source of error, however, is constantly guarded against. The sampling error has varied from 8 per cent to 20 per cent in this class of ore.

Samples are also taken from the various ore bins on the 1000 level or Hopewell haulage tunnel.

The stope samples are checked against the ore-bin samples, and the latter are checked against the smelter assays. In all cases when a discrepancy exists, the stope samples are higher than the ore-bin samples, which in turn are higher than the smelter samples. For the year 1928 the average error for all classes of ore between the stopes and the smelter was 5 per cent.

More specifically the methods and requirements of, and the duties of a sampler underground and in the office are as follows: On entering a stope or heading, the sampler determines the number of samples necessary and records the location of the advance (measured with a tape from known points) in his notebook; he then marks the sample numbers plainly with heavy white chalk on the working face. The marks are made heavy enough to remain on the walls until the sampler on the night shift returns. The sample tags are then filled in,

showing the location of the sample with regard to a known point, after which the sacks are distributed, and the samples taken.

If slips, dikes, or schistosity are parallel to the face, the sample is taken across the back of the floor below, covering the same area. When the sampler can not get under the back, any dikes or slips occurring on the face are broken into by plugging to permit sampling of the ore. Where it is impossible to obtain a satisfactory sample by the above methods, a series of drill holes, 5 to 10 feet deep, is drilled in the face and the sludges caught and used as the sample.

The samplers are held responsible for ore missed because of areas not being properly marked. The faces or walls must be marked with very heavy chalk either "O. K." or "N. G." The marks are heavy enough to last several days under ordinary circumstances; if the ore is not mined in that time, the faces are remarked.

The sampler is required to make trips to the shaft with samples at regular intervals throughout the shift. This permits the assay office to make determinations and to prepare the daily assay sheet before the night shift goes underground. Sample sheets with the daily stope sample returns are supplied to the foremen and shift bosses of both the day and night shifts. This enables the night-shift bosses to mine according to the results of sampling on the preceeding day shift and prevents the mining of much low-grade or questionable material.

Cooperation with the mining department is essential, and is secured very successfully. The samplers keep in close touch with the foremen and shift bosses whose ideas regarding the value of doubtful ore are very acceptable, and are acted upon immediately by the samplers; the sets or areas in question are carefully checked and rechecked, if necessary. Close contact with the geologists and their stope files enables the samplers to anticipate the general characteristics, peculiarities, and irregularities of the ore bodies.

The samplers keep two sets of assay maps, one in the general office for use by the superintendent and geologist, and the other in the foremen's office for the use of the division foremen and shift bosses. These books are carefully kept and show projected ore outlines, pillar and fence lines, chutes and raises, and the geology. They are posted every day in order to be up to date at all times. Thus the progress of any stope may be followed by watching these maps.

Development work is similarly plotted and posted after every round.

Some stopes are mined by means of vertical holes, and large areas are shot down at one time. In such cases, back samples are taken ahead of the cut, and mining is governed accordingly.

Occasionally samples of broken ore in the stopes, car samples, or chute samples are necessary. It has been determined that for such sampling the proper size to choose, as far as possible, is a piece 3 1/2 inches in all dimensions. Several cars of ore of approximately 1 ton each were chosen from various localities and sorted by hand into different sizes: Minus 1 inch; plus 1 minus 3 inches; plus 3 minus 5 inches; plus 5 minus 7 inches; plus 7 minus 10 inches. Each size was weighed and assayed, and the average assay for each car determined. From the assays it was found that the plus 3 minus 5 inch size was nearest the average for the whole car, and the plus 1 minus 3 inch size second best. From this it was deduced that the proper size to choose must be slightly over 3 inches in diameter.

Estimation of Tonnages and Values

All underground ores are divided into four classes, according to the gangue: (1) Massive sulphide (sulphide ores with less than 25 per cent silica); (2) silicious massive sulphide (sulphide ores with more than 25 per cent silica); (3) black schist; and (4) quartz porphyry. The proportions of these various types of ore vary in different sections of the ore gody and from level to level. Over the whole ore body the proportions are approximately as follows:

	Per cent
Massive sulphide	44
Silicious massive sulphide	34
Black schist	11
Quartz porphyry	11

The classification "ore blocked out" is used only under the following conditions:

1. Ore between levels when silled both above and below.

2. Ore silled on either top or bottom, with a raise finished and with drifts or crosscuts on top or bottom.

3. Ore with drifts and crosscuts on top and bottom, and with a raise finished, and with more than one diamond drill hole intermediate between levels.

4. In general, all ore where there is practically no risk of failure of continuity.

The classification "probable ore" includes:

1. Ore cut by drifts or crosscuts, but without intermediate work.

2. Ore silled on one level, but without workings of any sort on the level above or below, distance up and down not to exceed width of stope.

3. Ore cut by drifts or drill holes when within the ore zone may be taken up and down a distance equal to the width of the ore.

4. In general ore where there is a risk, yet a warranted justification for assuming continuity.

"Possible ore" includes ore which can not be included in either of the above classifications, nor definitely known or stated in terms of tonnage. This includes ore cut by diamond drill holes outside the main ore zone where good information is lacking.

Tonnage factors in terms of tons per cubic foot are computed from specific gravities. Samples of each class of ore from each stope or ore area are combined into composite samples and the specific gravities of these composites are used in calculating the tonnage factors for the respective classes of ore. In some cases composites from more than one floor are made; the averages of the specific gravities are then used in determining the tonnage factors. If no composites are available, the average for the mine is used. Representative tonnage factors are:

	Tons per cubic foot	Cubic feet per ton
Massive sulphide..................	0.1264	7.911
Silicious massive sulphide	.1030	9.709
Black schist............................	.1101	9.083
Quartz porphyry.........................	.0967	10.341

Three general cases are considered in determining volume. Where the ore continues from level to level, the formula for a truncated cone or prism is used; that is, Area of top + area of base/2 x H = Volume. (H = vertical height). If the ore pinches out to a line, the wedge formula is used; that is, A/2 x H = Volume. Where the ore pinches out to a point, the cone or pyramid formula is used; A/3 x H = Volume. Wherever it is known that the ore pinches or swells between levels, the volume of the ore body is figured in two or more blocks, being split where the dip of the ore changes. In general, the volumes of definitely outlined but irregularly shaped blocks, such as those that contain pillars, are calculated as special cases, adhering to geometric principles.

A correction for dike dilution of both grade and specific gravity is used for all dikes over 1 foot wide. In fire areas, where the ore limits can not be sampled or accurately determined, but are surrounded by old stopes, the wall assays of which show ore, 50 per cent of the area is assumed to have the average value shown by the old assays and 50 per cent of the area to average 3 per cent copper. This, in a rough way, includes those samples approaching our mining limit of 3 per cent copper.

In the calculation of grade, all stope samples are used, making adjustment by the use of wall assays according to the method or direction of mining. Erratic high assays are reduced to the general average of adjacent samples unless local conditions indicate otherwise.

In estimating ore reserves, a detailed knowledge of the principal ore deposit of the mine and the relationship of the smaller ore bodies is requisite. The method of projecting ore outlines is empirically chosen in each case, and therefore can not be stated.

DEVELOPMENT AND MINING

Development

All exploration, development work, stoping methods, tramming, and hoisting equipment are planned with the definite purpose of providing and maintaining an underground production of 3,000 tons of ore per day. To do this it is necessary to develop four new levels every five years, drive 20,000 feet of drifts and raises, and do 25,000 feet of diamond drilling yearly.

The ore body at the United Verde has a definite rake to the northeast, so that in developing new levels in depth the general location of the ore is known. More detailed data prior to laying out the haulage drifts to the ore body is secured in one of two ways - namely, by diamond drill holes drilled down from the lowest level, or by a winze sunk on the sulphide-schist contact, permitting some development during the period of extending and concreting the main shaft. Knowing the general position of the ore body, the main drifts from the shafts are laid out to intersect the center of the ore body, which is then developed as

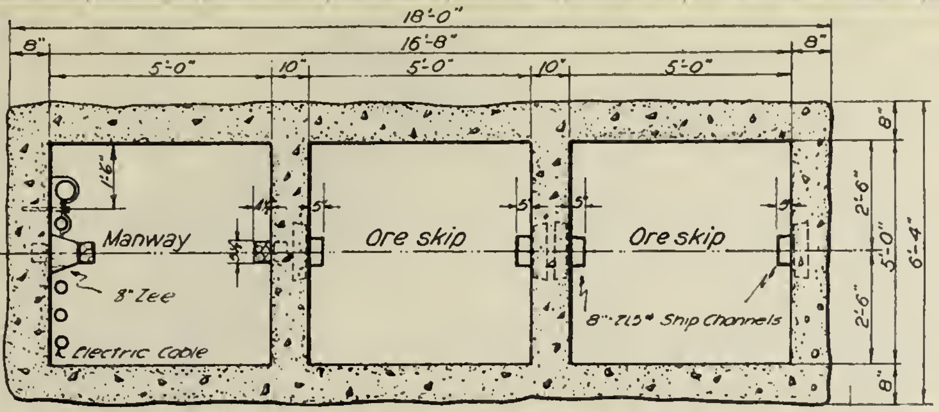


Figure 4.- Horizontal section through No. 5 shaft

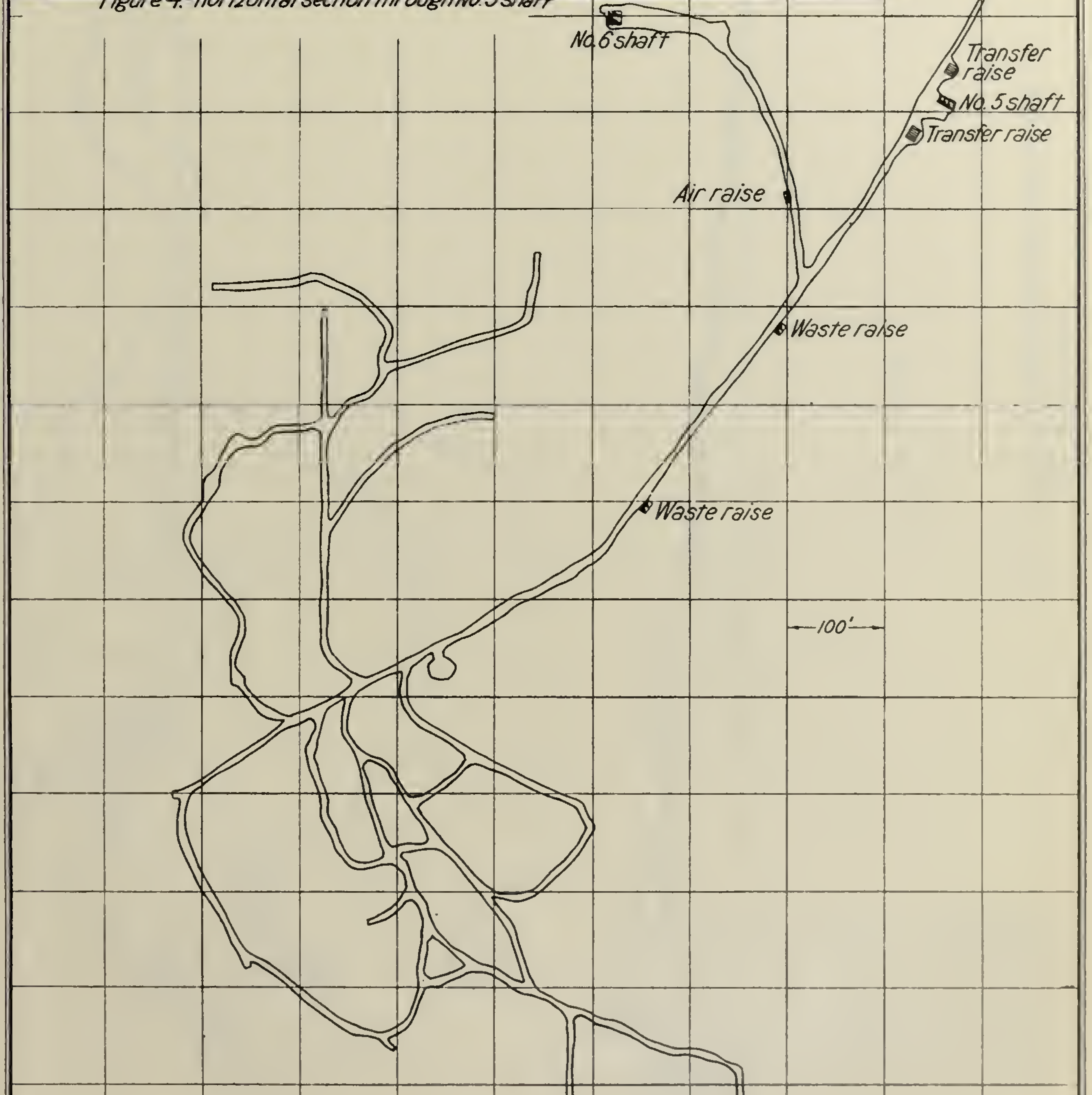


Figure 3.- Plan of a typical level

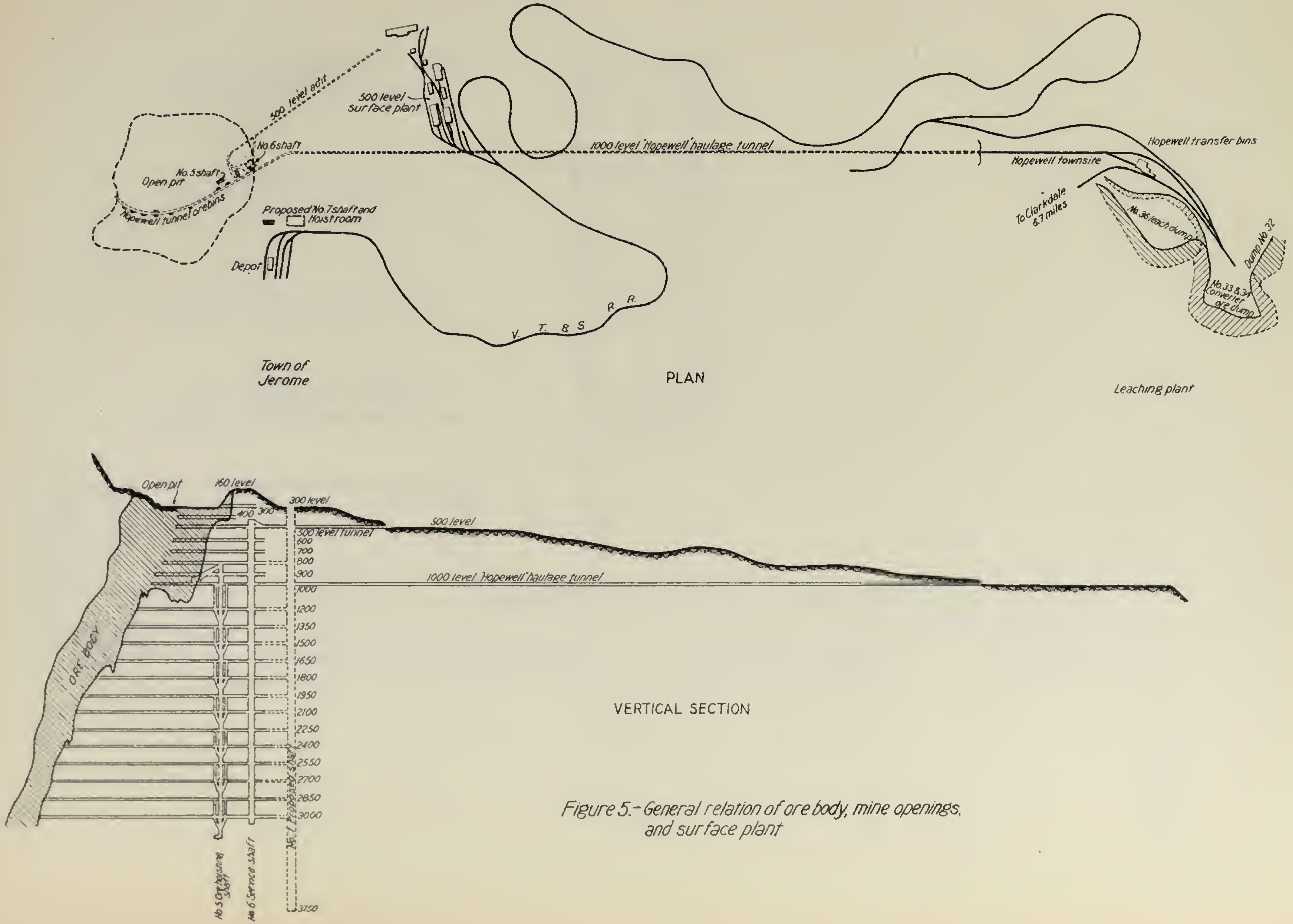
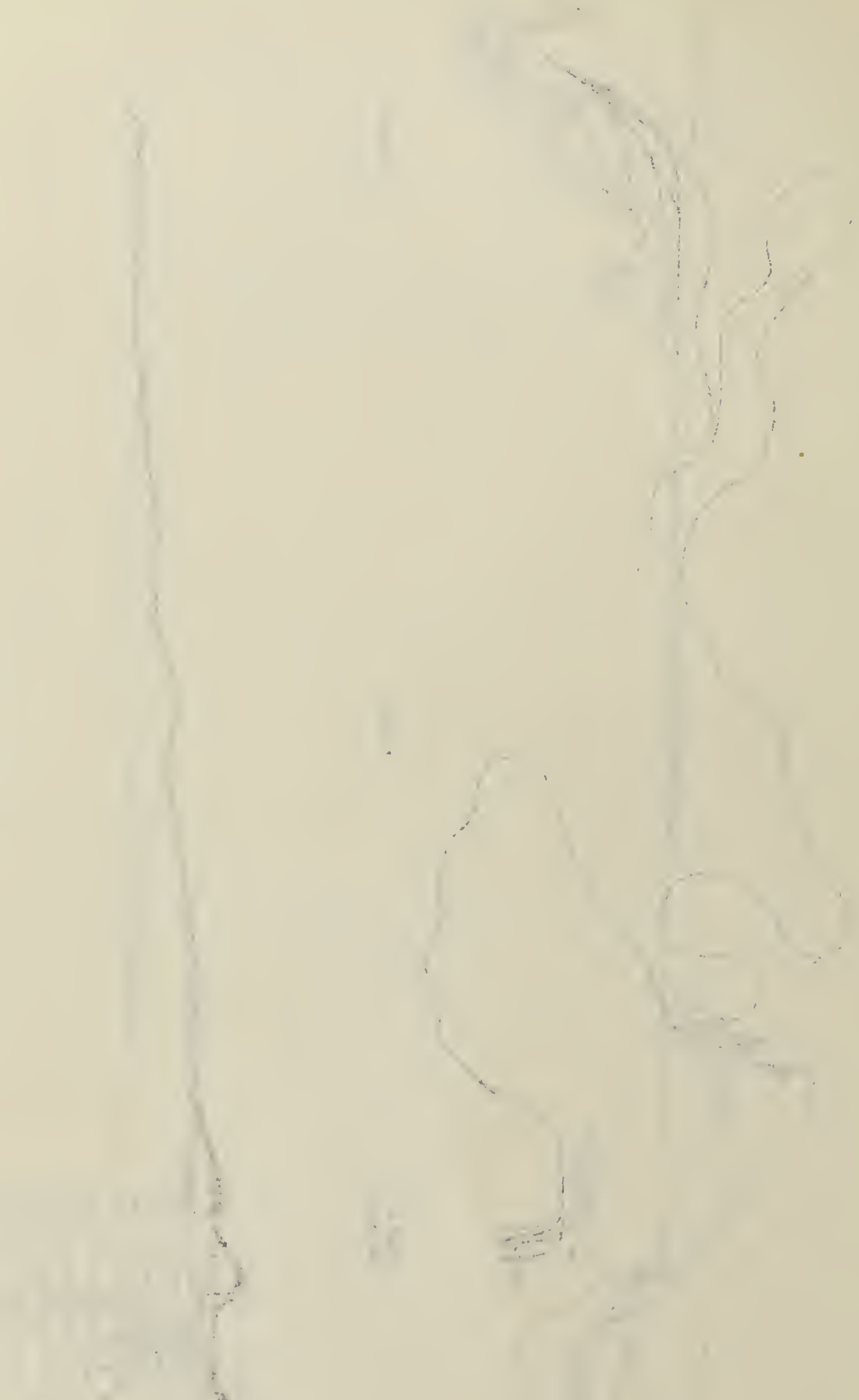


Figure 5.- General relation of ore body, mine openings, and surface plant



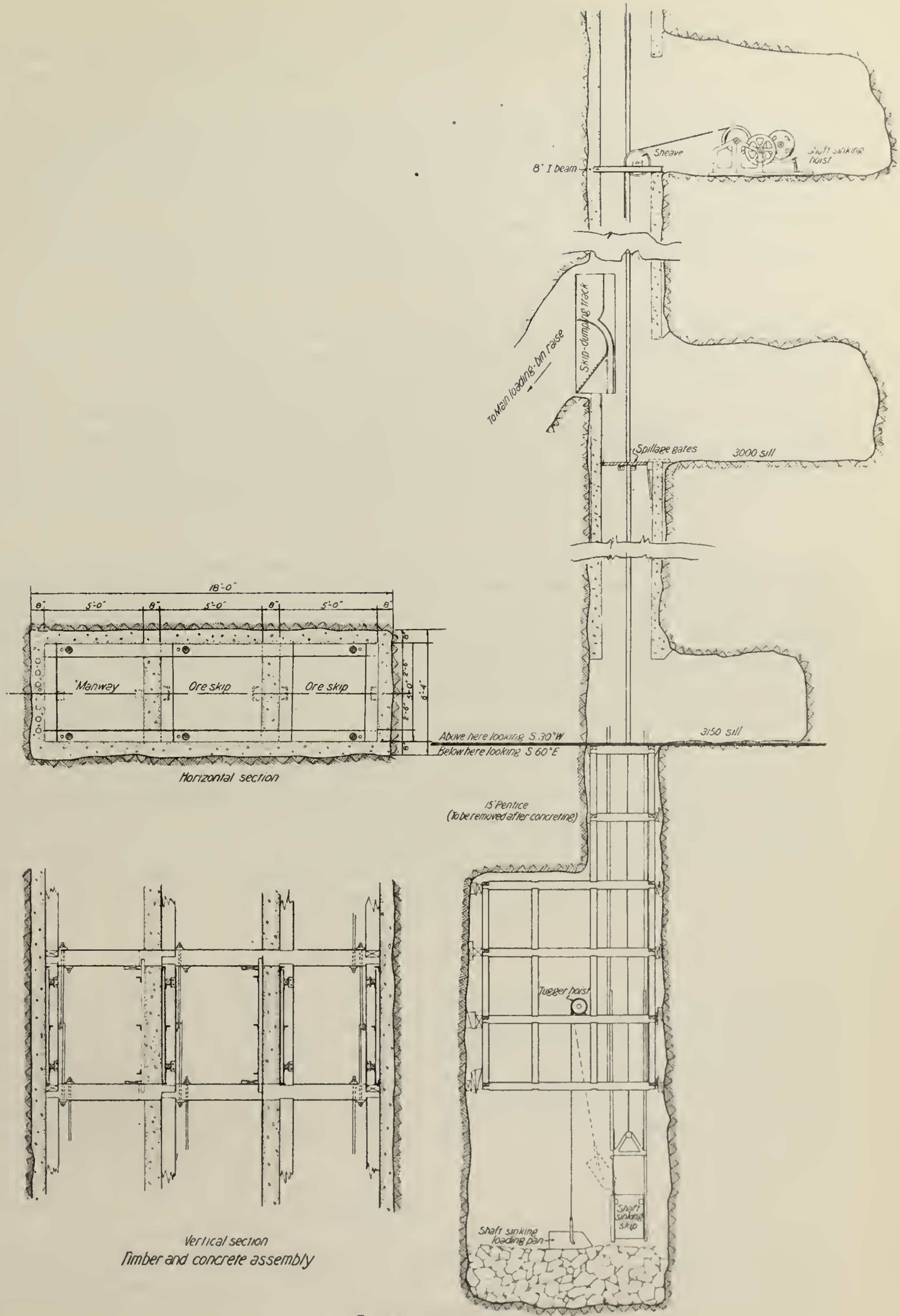


Figure 6.- Shaft-sinking layout, No 5 shaft

described under stoping methods. Figure 3, a plan of a typical level, shows the location of the two main shafts with respect to the ore body. Figure 5 shows the general relation of ore body, surface plant, and mine openings.

The mine is served by two main shafts, both of which have underground hoist rooms and are of reinforced concrete construction. No. 6 shaft is used exclusively for supplies and men. It extends from the 500 to the 3000 level, which is the bottom level at the present time, and is connected to the mine surface plant by a 1,600-foot adit. The cage has two decks, 6 feet 9 inches wide by 12 feet 7 inches long, and is double-tracked, permitting timber and steel to be delivered to the various levels with no transfer of materials. The capacity is 100 men, or eight standard trucks.

No. 5 shaft is the main hoisting shaft. All ore from above the 1000 level is dropped by gravity through ore passes to loading bins on that level. Ore from below the 1000 level is hoisted and delivered to similar bins on the same haulage level. Ore from the open-pit operations is also dropped to this level through raises. From here all ore is transported through the Hopewell tunnel by standard-gage, 40-ton cars for a distance of 1.8 miles to outside bins. From there the mill and direct-smelting ore is shipped to Clarkdale over the Verde Tunnel and Smelter Railroad. Low-grade silicious ore, leaching ore, and excess waste from the shovel operations are delivered to dumps by 25 cubic yard dump cars.

Mining operations extend from the 300 level, at an elevation of 5,207 feet, to the 3000 level, at an elevation of 2,528 feet. The sulphide mass above the 900 level has been bulkheaded off because of fire, which started in 1894. The greater portion of the ore above the 600 level will be mined by shovel methods. Border ore outside of the pit limits and fire-country ore above the 900 level will necessarily be mined by underground methods after the pit has cooled the country sufficiently to permit mining operations. For this reason all levels must be maintained and kept open.

Above the 1000 level, the level interval is 100 feet. One 200-foot lift from the 1000 to the 1200 level proved this interval excessive because of chute repairs and increased cost of raise development. Below this horizon the level interval is 150 feet. Formerly, every second level was planned as a trolley motor-haulage level. Recent practice has been to haul the ore to the shaft on every level with storage-battery locomotives. This is more economical than passing the ore through transfer raises to the level below.

No. 5 Shaft

No. 5 shaft, Figure 4, is used exclusively for handling ore. It extends from the 800 to the 3000 level, and is now being sunk to the 3600 level. It is of reinforced concrete construction, with two 5 by 5 foot hoisting compartments, and a 5 by 5 foot manway and pipe compartment, which is also used in sinking operations. In the lower part of the shaft 8-inch, 21-1/2 pound ship channels are used for guides, and in the upper portion of the shaft 3-5/8 by 7 inch seasoned Oregon pine with a wearing strip of 1 by 3-5/8 inch clear white oak is employed.

Shaft Sinking

Figure 6 shows the general method of sinking No. 5 shaft. A 15-foot pentice is left below the bottom level by sinking on the manway side and widening out bolow. Tho shaft is timbered with 8 by 10 inch Oregon pine shaft sets, so designed as to permit concreting,

using 2-inch plank panels for forms. A 32-hole round is used, and the cut placed to take advantage of the ground. The rounds are fired by No. 1 to 10 electric delay detcnators using 440-volt alternating current. The crew consists of three shaftmen, one leader, one top man, and one hoistman. The average standard on which a bonus is paid varies from 0.17 feet per miner shift at the beginning of a lift to 0.10 feet at a depth of 600 feet below the bottom level. The following table shows the cost of explosives and labor:

Sinking data, No. 5 shaft

Drilling speed, inches per minute	10
Number of holes	32 to 35
Advance per round, feet	4.5
Powder used, sticks	260
Powder cost per foot advance	$5.20
Labor cost per foot advance	$31.85

The explosive cost does not include fuse or detonators. The labor cost includes timbering and mucking. The water encountered is not serious and averages around 1,000 gallons per hour.

Former practice has been to use a 30 cubic foot sinking bucket, with the usual auxilliary equipment. Present practice is to shovel into an 18 cubic foot pan, which is then raised by means of a small air hoist and dumped into a 38 cubic foot skip. The skip has extended guide shoes to permit lowering it to the bottom of the shaft. The skip dumps automatically into a "dog-hole" leading down to the main skip-loading pocket. The pan is much easier to shovel into than a bucket, and the skip eliminates a second handling on the level above. In addition, broken rock from cutting loading-pockets can be drawn into the skips by gravity.

The work of concreting the shaft is started from the bottom, using 2-inch plank panels for forms. The central mixing station is located on the 500 level, and the concrete is delivered through a 5-inch pipe line. Six hundred feet of shaft was concreted in five months, using 21 men on three shifts. The total cost of concrete per cubic yard was $22.50.

No. 6 Shaft

No. 6 shaft is used only for service (see fig. 7). It extends from the 400 to the 3000 level. Above the 1950 level it is concreted solid, but below 2-1/2-foot concrete rings are placed at 6-foot centers. The cage compartment is 8 by 13 feet, the manway compartment 4 by 9 feet 4 inches and the counterbalance compartment 4 by 3 feet. A sublevel at the collar of the shaft on the 500 level permits loading and unloading of both decks at one time when handling the shift. The shaft is raised from each level in full section, using a double cribbed manway. Concreting is started at the top and carried down as the broken rock is pulled from below. Sectional steel forms are used, supported on 4 by 10 inch sills by means of adjustable hooks from the last set poured. Eighteen men on three shifts will draw the broken rock, trim the walls, set forms, and pour one ring every three shifts. A comparative cost of the solid concrete and the ring methods of construction for No. 6 shaft is given in the following tabulation.

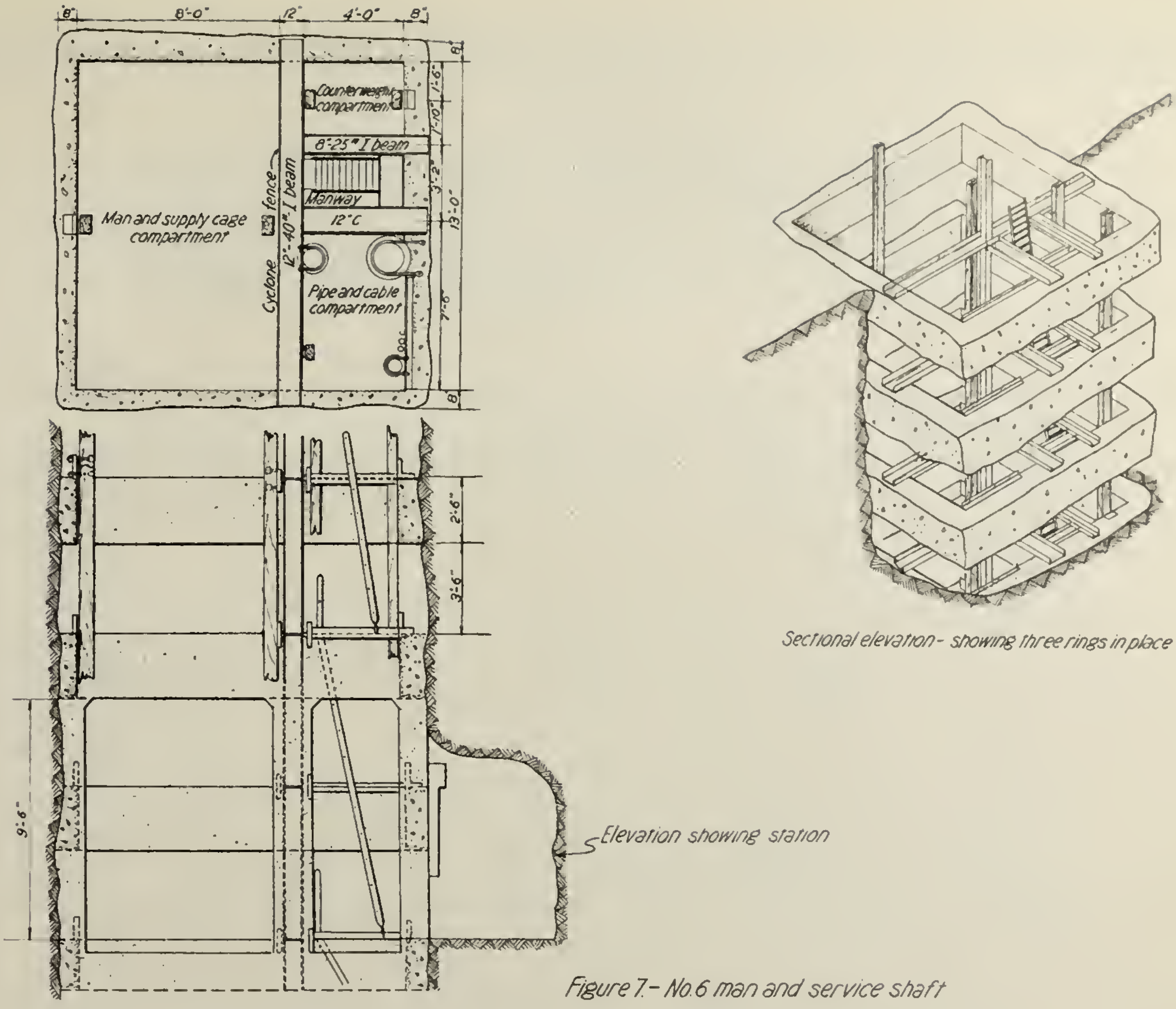


Figure 7.- No.6 man and service shaft

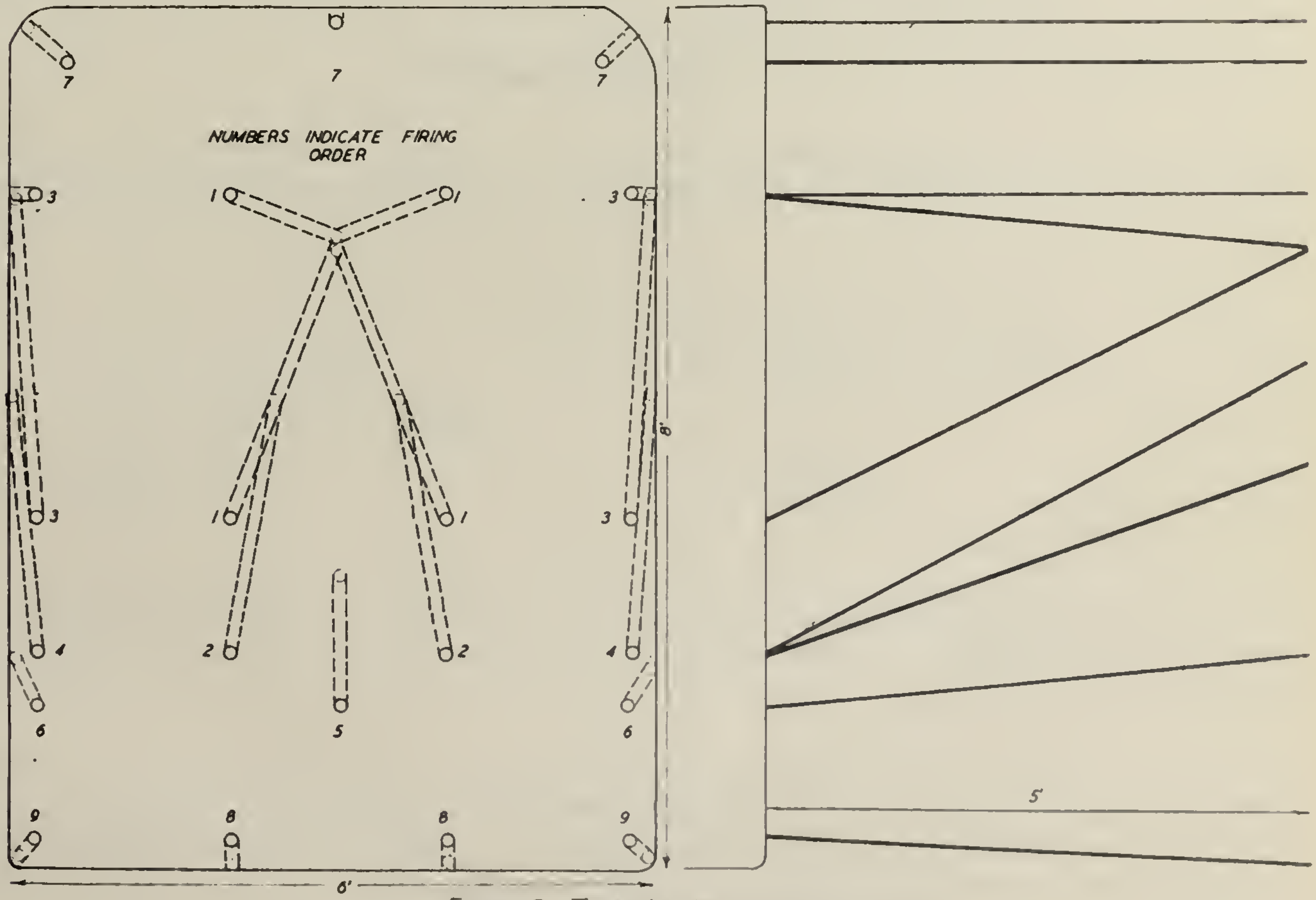


Figure 8 - Typical drift and crosscut round

Comparison of cost of solid concrete and concrete-ring methods of shaft construction

Method	Concrete mix	Feet	Excavation per foot	Concrete per foot	Total per foot
Solid	1 : 3 : 5	1,550	$68.17	$146.84	$215.01
Ring	1 : 2 : 4	1,050	77.11	50.11	127.22
Saving per foot by ring method					87.79

Detailed costs -- concreting by ring method	Excavation per foot	Concreting per foot
Labor	$52.07	$16.60
Shops	0.26	5.90
Supplies	7.68	26.51
Engineers	-	1.05
Explosives	8.47	-
Compressed air	2.08	-
Repairs	6.55	-
Miscellaneous	-	0.05
Total	$77.11	$50.11

Cost of concrete per cubic yard poured Solid = $61.30; Ring = $58.80.

The above costs do not include steel work, ladders, or pipe lines.

Drilling and Blasting

There is such a wide variation in the class of ground encountered that drilling speeds vary from 4 to 48 inches per minute. The following is a tabulation of drilling speeds in the various formations, taken with a CP-6 drifter:

Rock formation	Drilling speed, inches per minute
Massive sulphide	4 to 8
Silicious massive sulphide	6 to 10
Black schist	24 to 48
Porphyry	10 to 30

The wide variation in drilling speed in the black schist results from the different degrees of alteration of the rock, whereas in the porphyry it is caused by the varying amounts of quartz.

The breaking qualities of all the rocks depend on many local geological factors and vary greatly. No definite statement can be made concerning any of them except that the black schist usually breaks quite fine. The breaking quality of the massive sulphide is apparently affected most by the copper content: The higher the copper content the finer the fragmentation. The siliceous massive sulphide seems to follow no rules. The porphyry, al-

though not the hardest rock, is undoubtedly the toughest and the most difficult one in which to get a good fragmentation except where it is altered or already broken by small slips or joints.

A 145-pound mounted drifter is used for practically all drifting and stoping. Several makes of stopers are in use for raising, both hand and self rotated. Two makes of light (38 and 52 pound) jackhammers are used for plugging and a 65-pound hand-held machine for sinking. All drills are of the wet type.

Three sizes of drill steel are used: 7/8-inch quarter-octagon for jackhammers and stopers in soft ground; 1-1/4-inch hollow-round for all Leyner-type machines; and 1-inch quarter-octagon for stopers in hard ground. The double-taper cross bit with 5 and 14° taper is standard for all steel. Starters are 30 inches long, with 1-7/8-inch gage. Length changes are in multiples of 10 inches, with 1/16-inch gage change. Columns and crossbars are made from 3-inch extra heavy pipe.

Two sizes of hose are used: 1/2 inch for water, and 1 inch for all air hoses. All hose is provided with standard lugged couplings and spuds which fit all types of machines and 1-inch pipe lines. Damaged hose is brought to the surface for repairs and mended by means of a compressed-air device, which cuts the hose, inserts nipples, and wraps the joint with wire. Another mechanical device forms the wire loops for wrapping joints.

Fifty per cent strength gelatin dynamite in 1-1/8 by 8 inch sticks is used in sulphide and porphyry, and 35 per cent in schist and for all plugging. A smooth, black-finish, cotton-countered safety fuse is used in 4 and 5 foot lengths for blockholing and in 7 to 9 foot lengths for development and stoping. No. 6 and No. 8 detonators are used with 35 and 50 per cent strength gelatin dynamites respectively. Electric blasting is employed in all shaft sinking and in drifts starting from shafts and winzes. Primers are delivered to the underground fuse magazines in wood-lined containers made of carbide cans and provided with hinged covers. Stemming consists of a clayey silt in paper cartridges and is used in all hard ground.

Bowlders are blasted during lunch hour, and stope and drift rounds are blasted only when the men are going off shift. An electric signal system between levels serves to eliminate the danger of blasting by a clearance signal from the level above. This signal system is essential because of the danger from sulphur dioxide and hydrogen sulphide formed by blasting in heavy sulphides.

Drifts and Crosscuts

Main drifts and crosscuts are carried 6 by 8 feet in cross section in untimbered sections and 5 by 7 feet where timbered. Gangways under stopes, when timbered, are carried 5 by 10 feet in the clear. When air conditions are conducive to decay, the timber is treated with zinc chloride. Only about 5 per cent of the drifts require timbering at the time of driving, and these are drifts in the area of the Verde fault and a few drifts in greenstone. Drift sets are usually made of 10 by 10 inch Oregon fir, but occasionally 8 by 8 inch or 12 by 12 inch is used, as required by the pressure.

The number of holes and type of round vary greatly, depending on the character and hardness of the ground. The rounds most commonly used are the V-cut, toe cut, and upper cut. In any of these rounds, the number of holes will vary from 10 to 28. Figure 8 shows a typical round for a 6 by 8 foot drift.

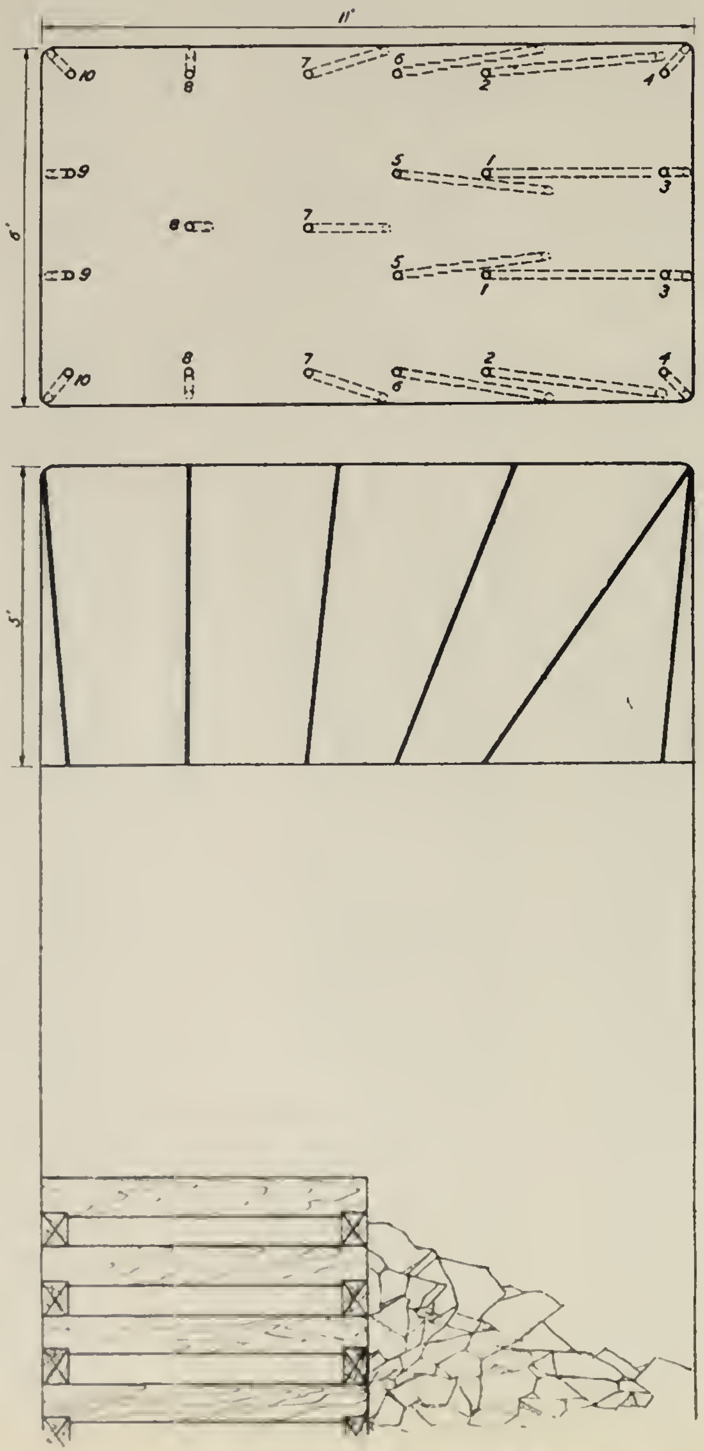


Figure 9.-Raise round. Numbers indicate firing order

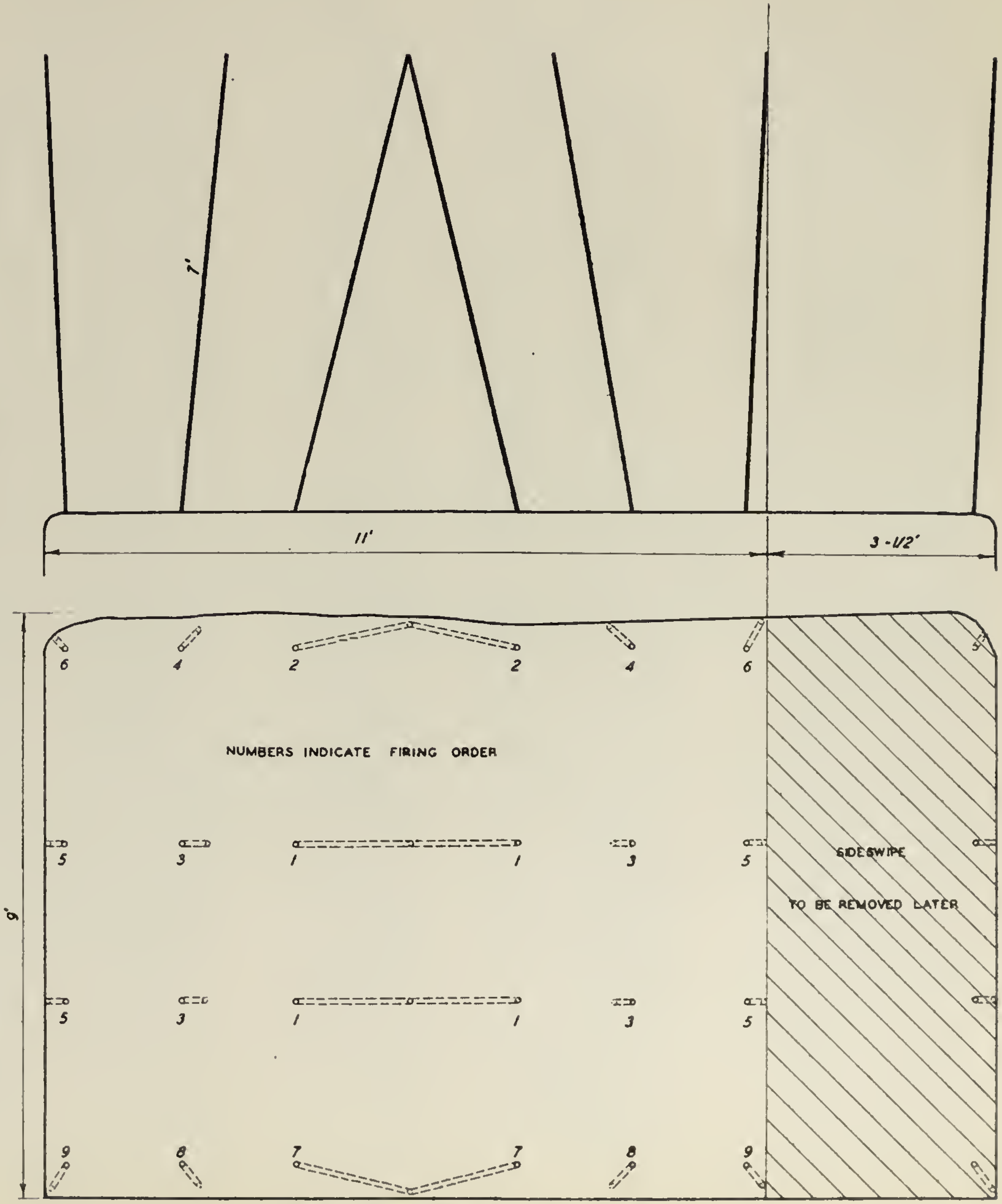


Figure 10.-Hopewell haulage-tunnel round

All drifts are worked two 8-hour shifts per day. A miner drilling with one machine, using a vertical bar, usually drills and blasts in one shift. Two muckers clean the drift on the opposite shift. In extremely hard sulphide it occasionally requires as much as four drilling shifts to get in a round. The average drift requires 136 sticks of 1-1/8 by 8 inch 50 per cent strength gelatin dynamite to the round. Drilling and breaking in drifts are done on contract, which includes labor and explosives. The mucking is done under the bonus system.

Raises

Raises between levels are 6 by 11 feet in section, and the manway side is carried up with single native-pine 5 by 8 inch cribbing with 56-inch inside clearance. The number of holes required depends on the character and hardness of the ground, but will average about 22. The cut is drilled over the chute side, as shown in Figure 9. Material is handled to the face by either a column-type Little Tugger or an Anaconda-type air hoist, used with a 12 by 14 inch by 4-foot steel bucket and 3/8-inch steel cable.

The miners are on contract; they drill and blast two rounds, then raise the cribbing to within 7 feet of the back. The regular motor crews draw the broken material on the opposite shift from the miners. Two miners work in each raise one 8-hour shift daily. The average round requires 144 sticks of 1-1/8 by 8-inch 50 per cent gelatin dynamite.

Adit

The main adit on the 500 level, which connects the surface plant to No. 6 shaft, is 1,600 feet long; 400 feet is untimbered and the remaining 1,200 feet is timbered with 10 by 10 inch Oregon pine sets, 8 by 9 feet in the clear, spaced on 5-1/2-foot centers. All timber and lagging have been partly covered with diamond-mesh wire and gunited for fire protection.

Hopewell Tunnel, on the 1000 level, is the main ore-haulage level, and connects the underground and shovel pit with the outside transfer and storage bins on the V. T. & S. Railroad. The adit is 10 by 13 feet in the untimbered portions; 2,700 feet is timbered with 10 by 10 inch Oregon pine, at 5-1/2 feet on centers. The sets are 9 feet high, with 10-foot caps; the posts are battered to give a 12-foot width at the sill.

The adit is now being extended and the heading is carried 10 by 11 feet and is untimbered. The width is limited by the amount of broken rock which may be gotten out in two shifts. The additional 3-1/2 feet will be "sideswiped." Figure 10 shows the round used. The 10 by 11 foot heading is drilled with two machines mounted on separate columns, the drilling and blasting being done on one shift. The round is cleaned out on the opposite shift with a mechanical loader. Approximately 136 sticks of 1-1/8 by 8 inch 50 per cent strength gelatin dynamite are used to the round.

Mechanical Loading

A portable loading dock, equipped with a double drum, column-type hoist, with a hoe-type scraper, has been used for loading in drifts, but has not proved entirely satisfactory. It is found more economical to load by hand in drifts which can be cleaned in one shift by two men. A compressed air operated mechanical loader is used in large drifts, pump sumps, hoist rooms, and occasionally in 6 by 6 foot headings.

Scrapers have been used to good advantage in stope sills where the chute spacing is irregular, and in spreading waste under favorable conditions. The two-drum air hoist has been replaced by a three-drum electric type, using two cables on the pull back. This permits the scraper to be moved in any position without constantly changing the tail block. Because scrapers under certain conditions tend to tear up the plank flooring, they have not been used to any extent in stoping operations.

Stoping

Although several mining methods are employed in the mine, over 85 per cent of the ore is extracted by two methods, namely--horizontal cut-and-fill and horizontal square-set. The other methods are used for exceptional conditions, such as in the recovery of level and vertical pillars and small, odd-shaped, isolated ore bodies.

The table below gives the percentage of ore extracted by the various methods in 1929:

Horizontal cut-and-fill	61 per cent
Inclined cut-and-fill....	3 per cent
Horizontal square-set....	25 per cent
Inclined square-set........	2 per cent
Shrinkage...........................	1 per cent
Top slice...........................	8 per cent
Total.................................	100 per cent

Horizontal Cut-and-Fill

In the development of a level, regardless of the stoping method to be used, a drift is driven along the sulphide-schist contact. The former practice of following the contact resulted in a very crooked drift, with difficult haulage conditions. As stated before, present practice is to drive in the schist, near the contact and roughly paralleling it by the use of easy curves and short tangents. From this drift the necessary raises and chutes are run for stoping operations. In some places the ore body is so wide that two parallel drifts are necessary to secure the proper chute spacing in the stopes. One of these parallel drifts is necessarily driven in the hard massive sulphide at a high cost per foot.

The practice at one time was to sill out either on the level or 10 feet above the track, and then put in a timbered gangway and chutes. This involved a high first cost for timbering and also a high maintenance cost, and was discontinued in cut-and-fill mining, although a somewhat similar procedure is still followed in square-set mining. The more recent practice is to drive an untimbered drift on the level and sill the stope at from 21 to 25 feet above the rail, depending upon the strength of the ground over the back of the drift (see fig. 11).

Due to the great variation in the width of the ore body and the irregularity of ore outlines, it is not possible to establish any fixed dimensions for the cut-and-fill stopes. They are all opened up to what is considered the maximum size consistent with safe mining conditions, both as regards the workmen and the physical condition of the mine. This varies from 30 to 160 feet across the ore body and from 60 to 200 feet along the strike. These dimensions are also affected by the necessity for maintaining regularity in the vertical pillar system (see fig. 12).

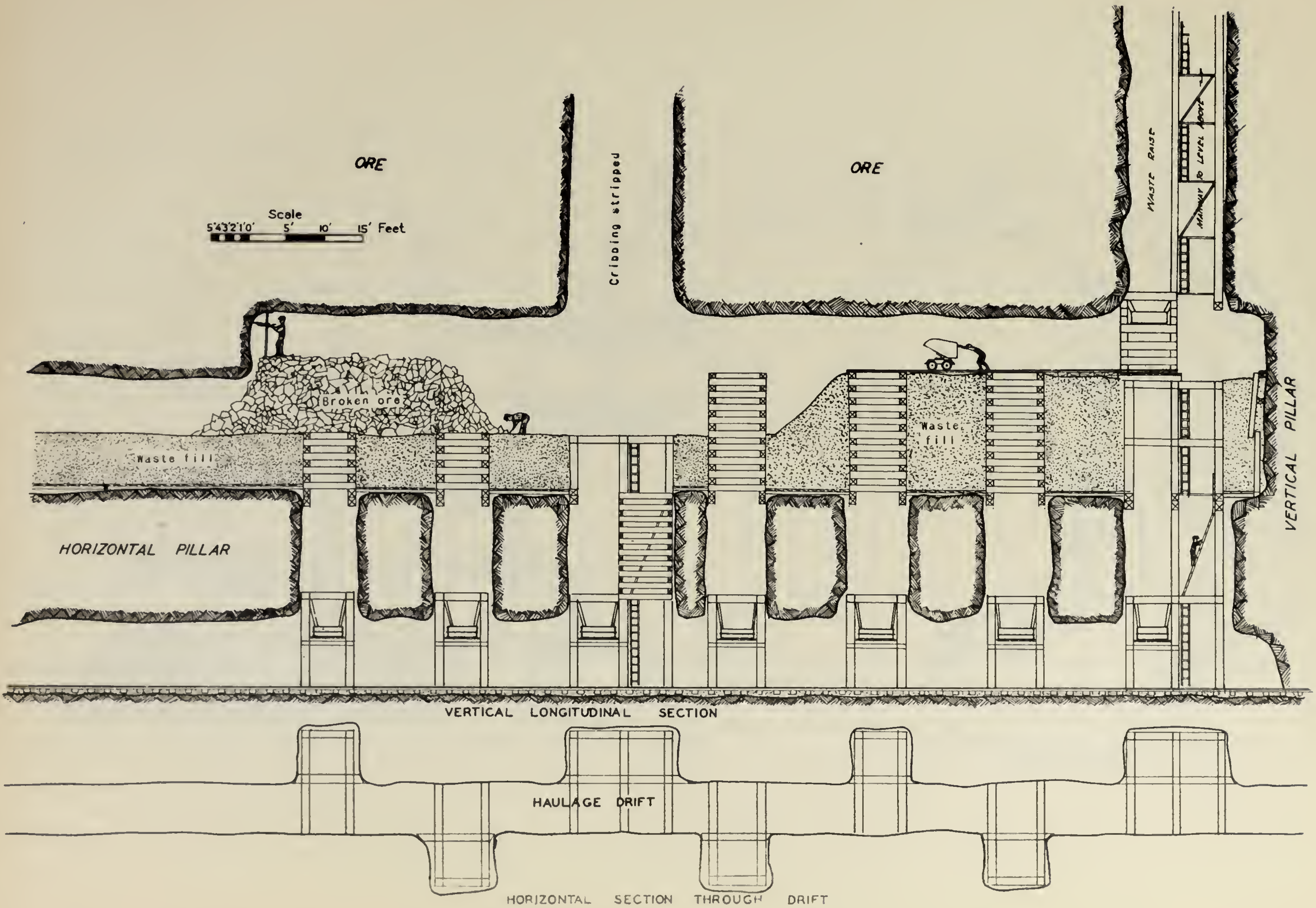
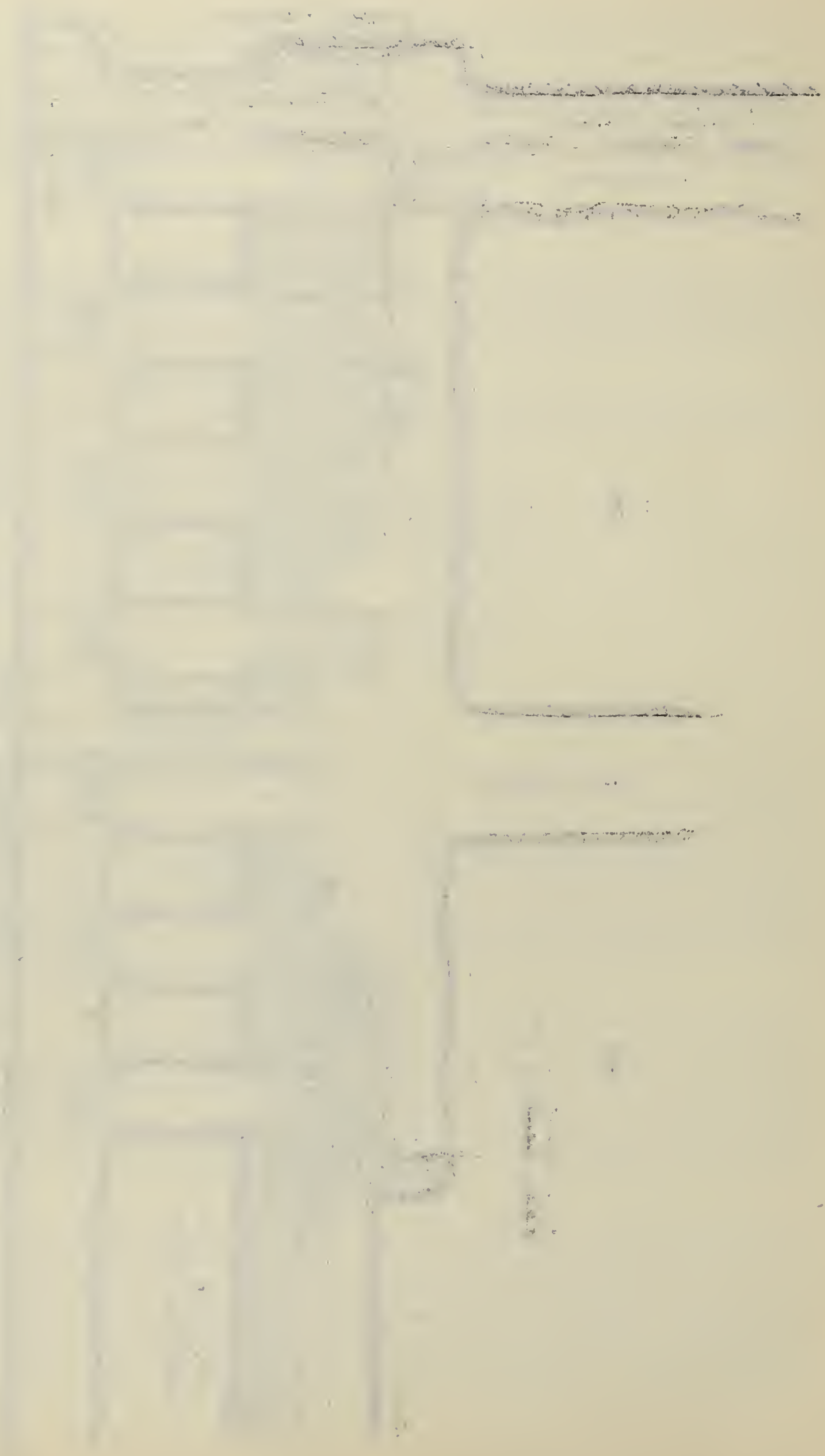


Figure 11 - Typical cut-and-fill stope



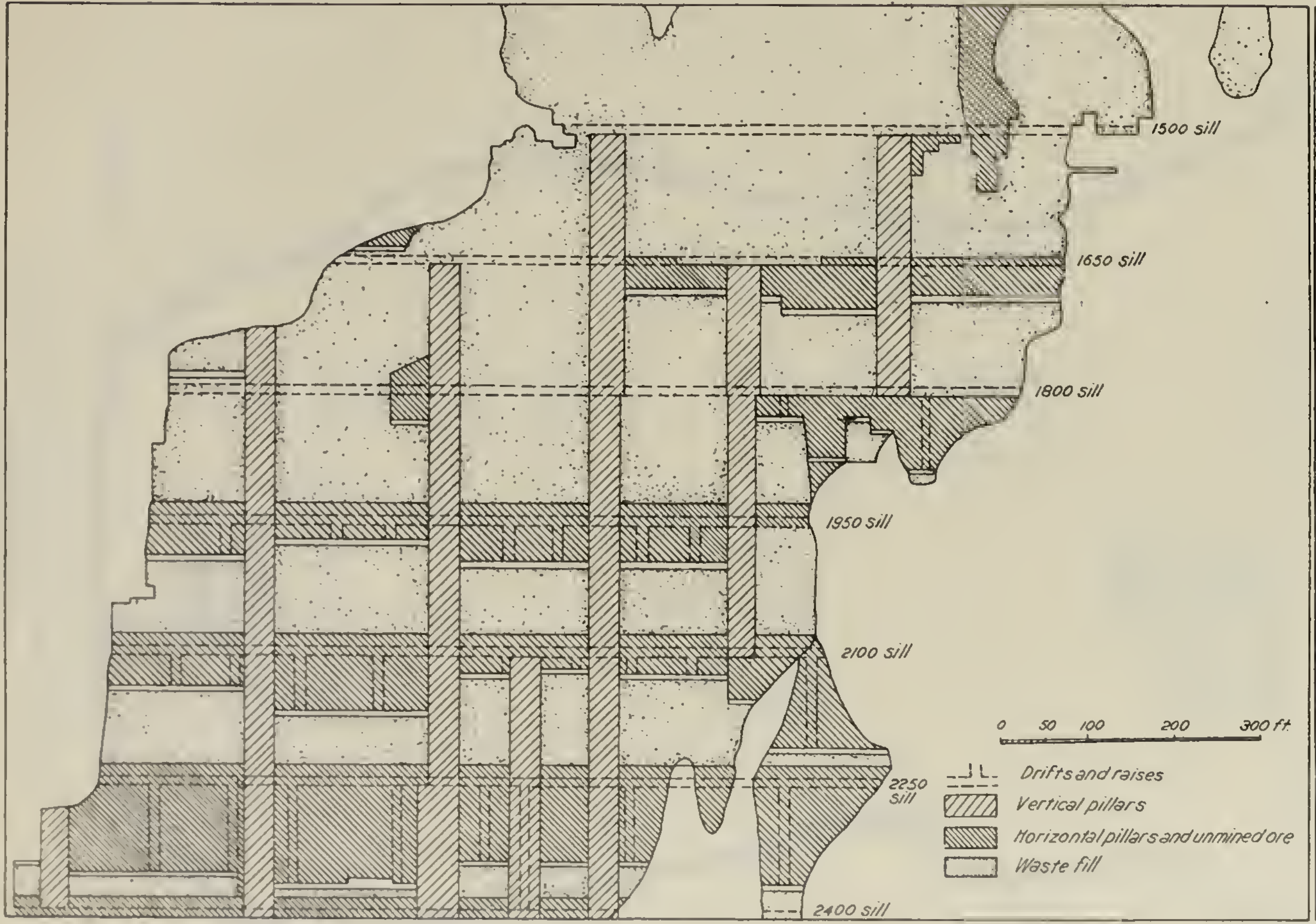


Figure 12.-Diagrammatic section 1500 to 2400 levels, showing pillar arrangement

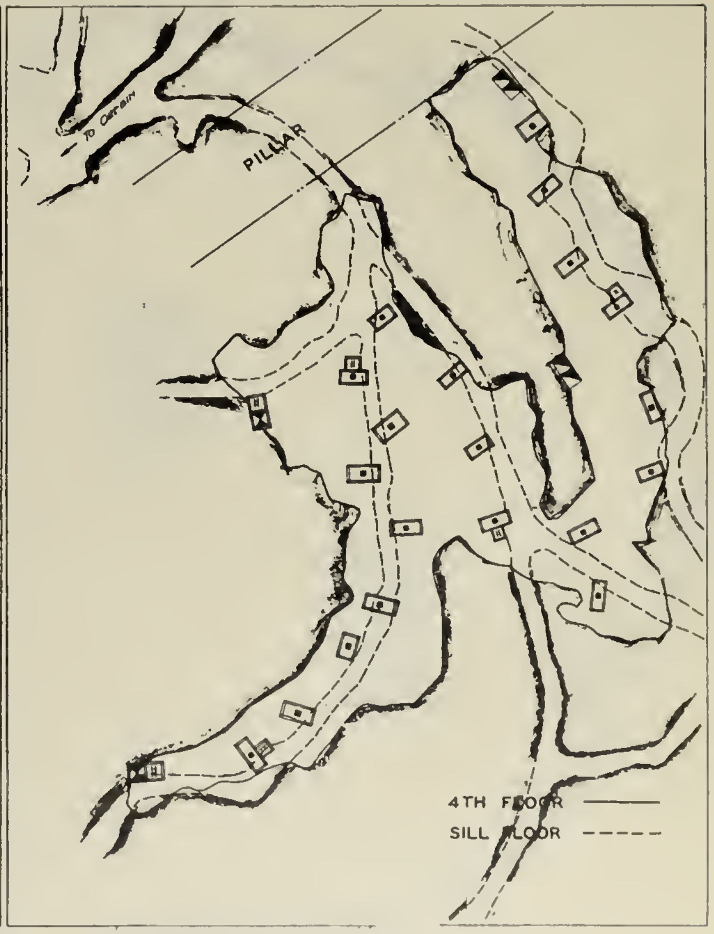


Figure 13.-Typical cut-and-fill stope, showing drift, chute, and raise layout

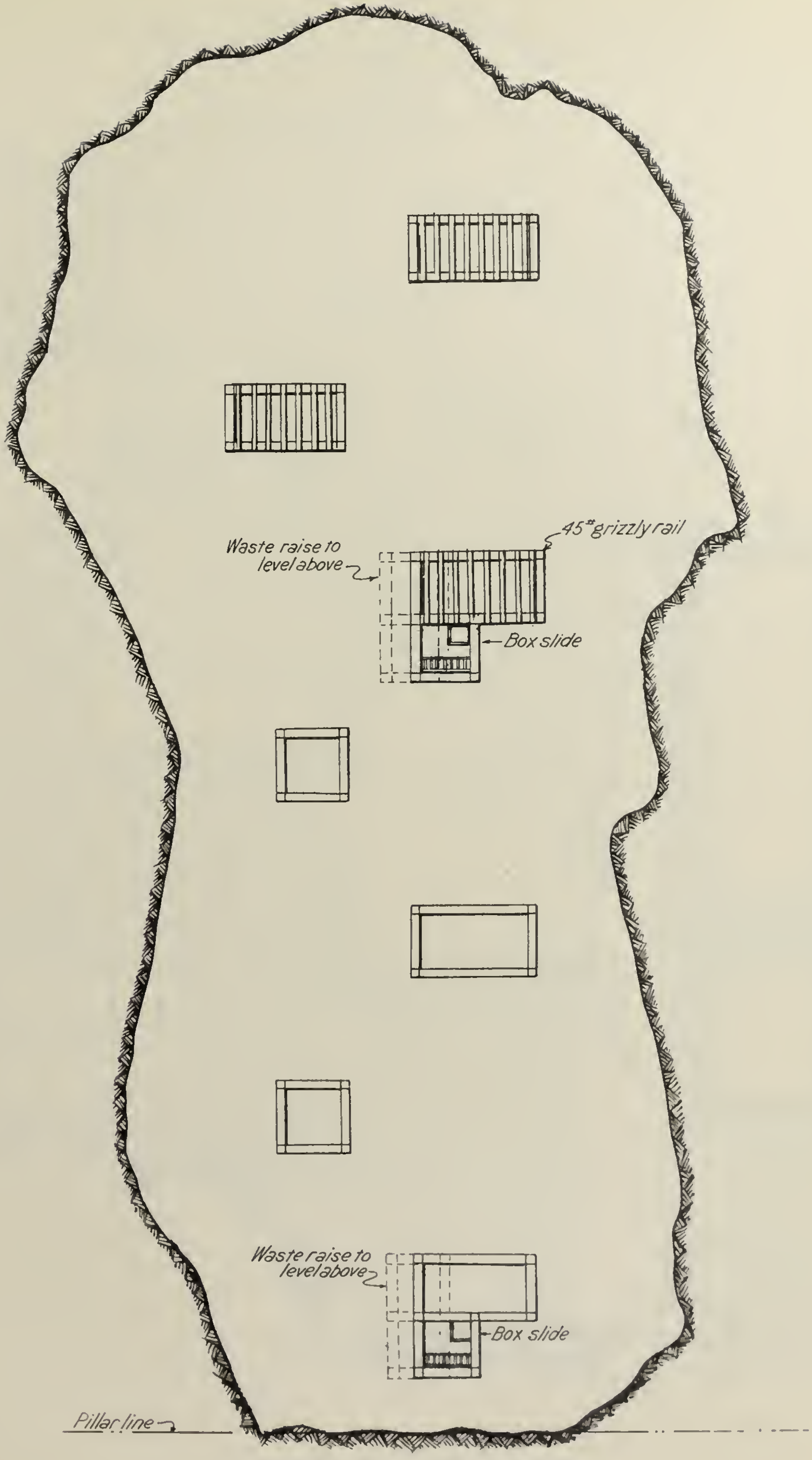


Figure 14.–Plan of typical cut-and-fill stope, showing chutes

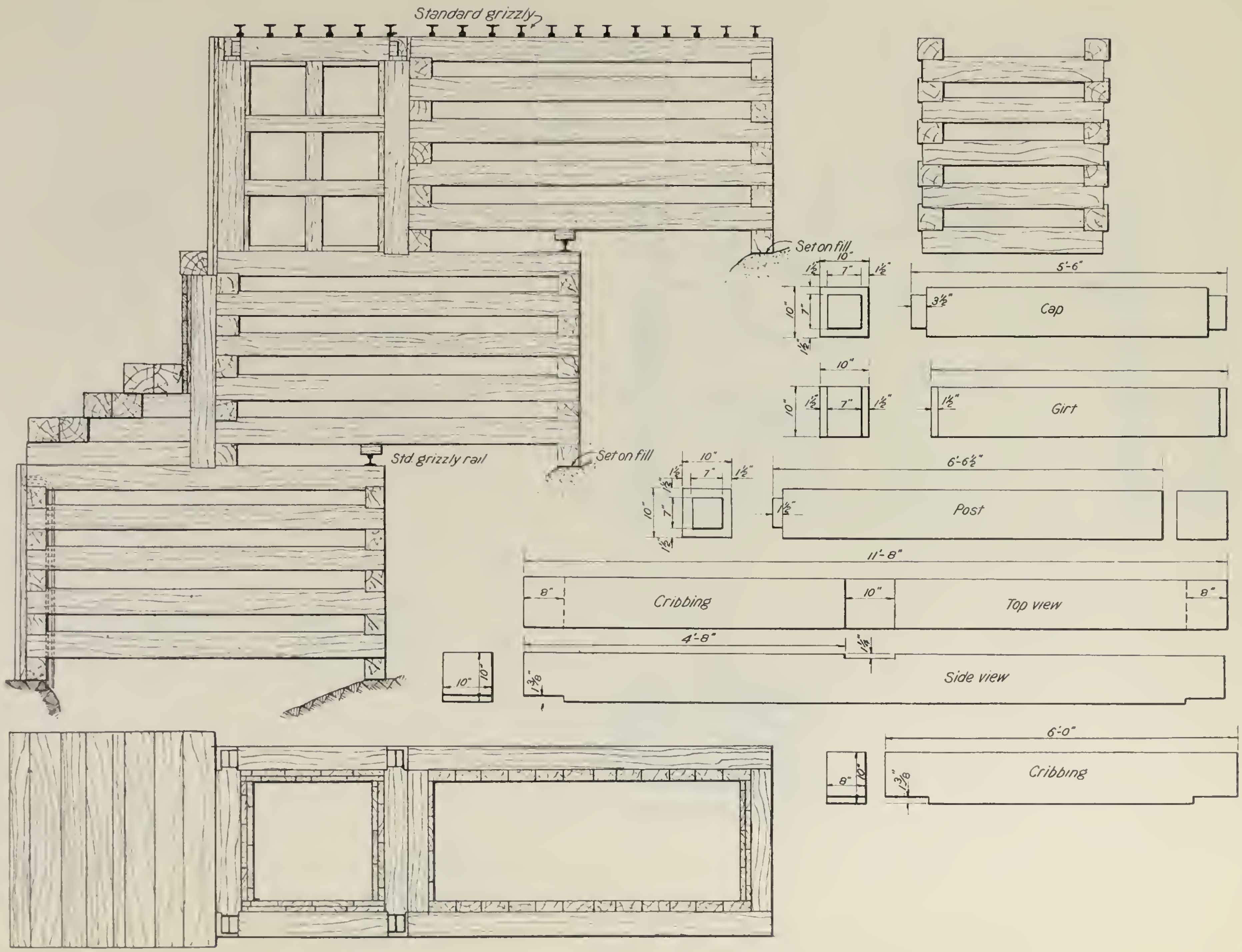


Figure 15.-Standard offset chute

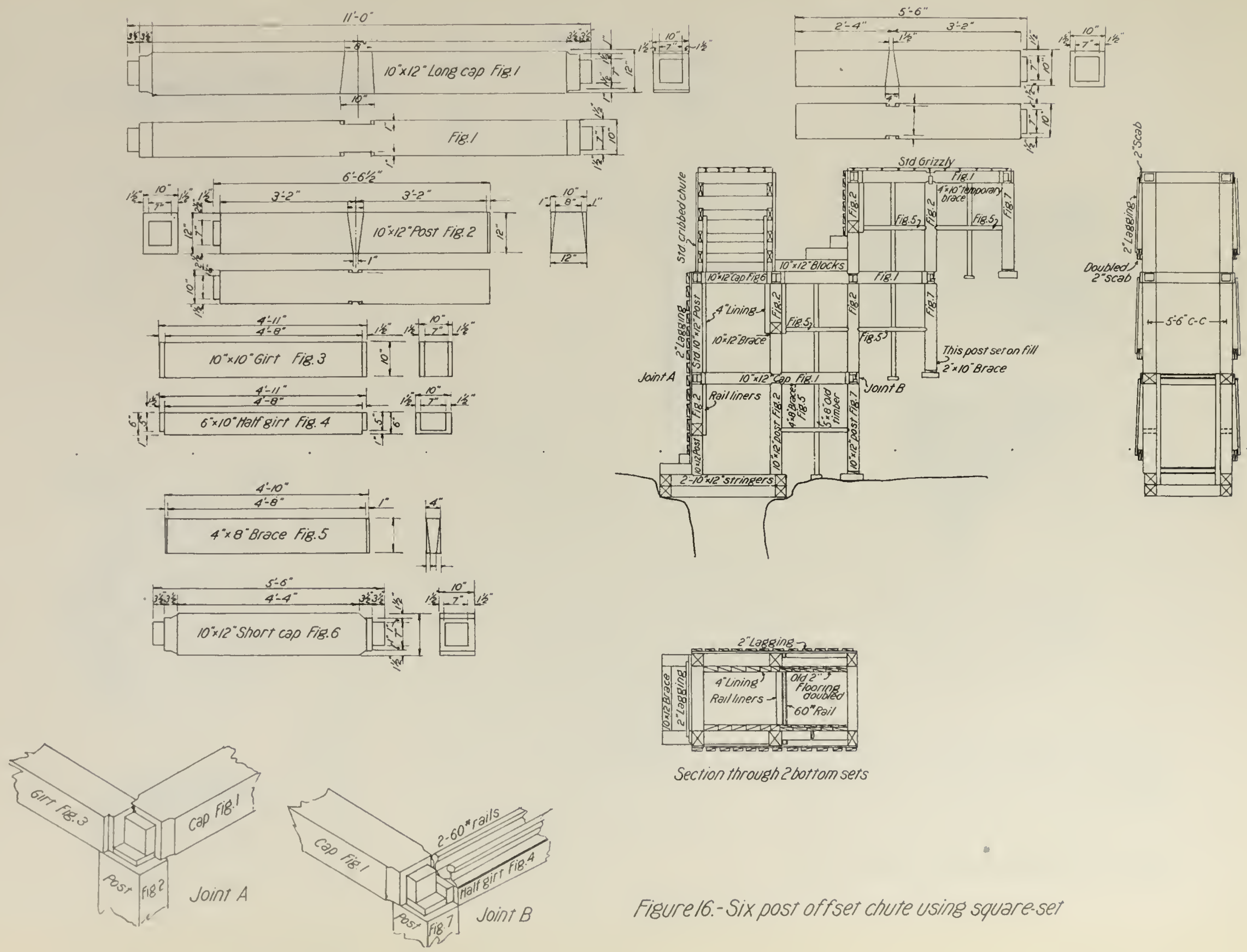


Figure 16.- Six post offset chute using square-set

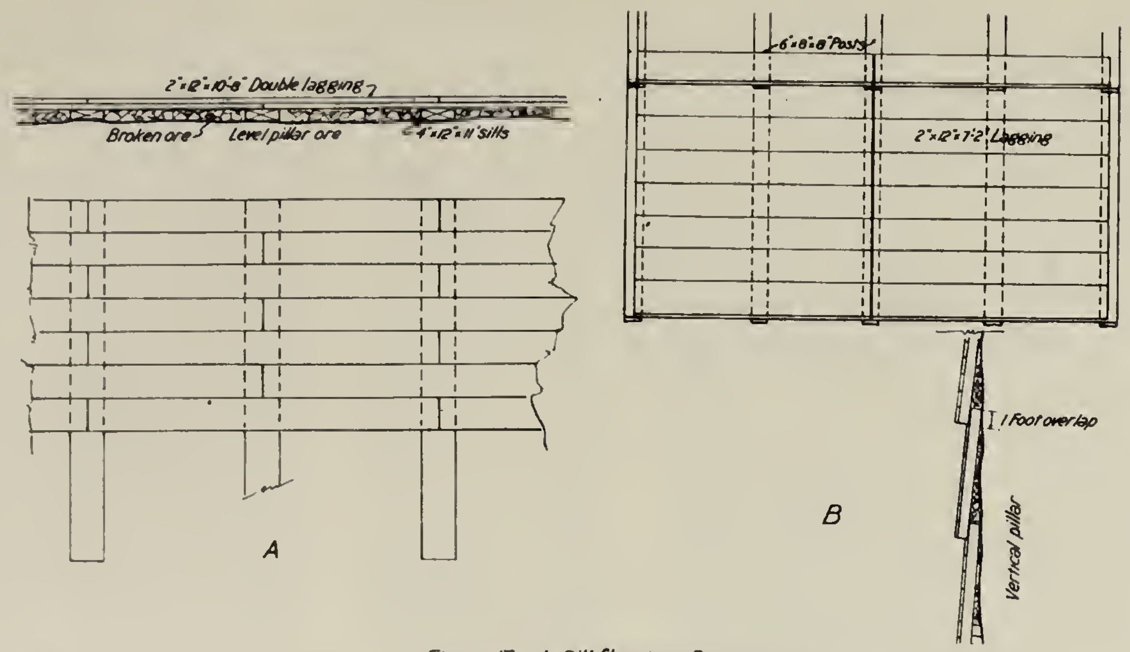


Figure 17.- A, Sill flooring; B, Pillar fence

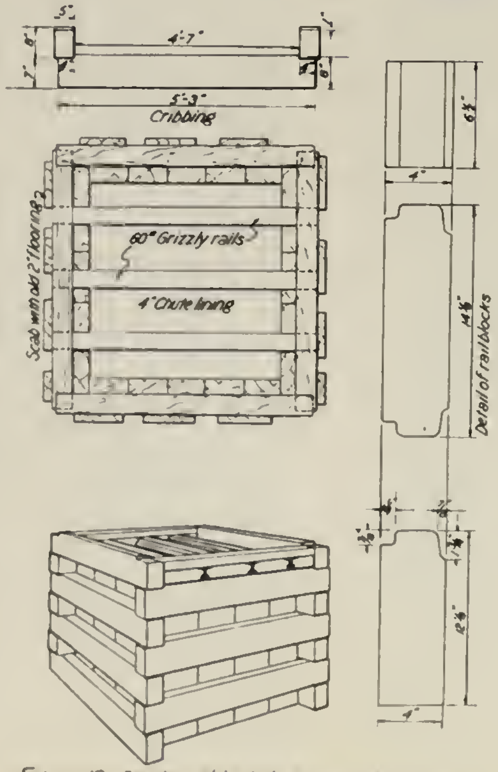


Figure 18.- Single cribbed chute and grizzly blocks

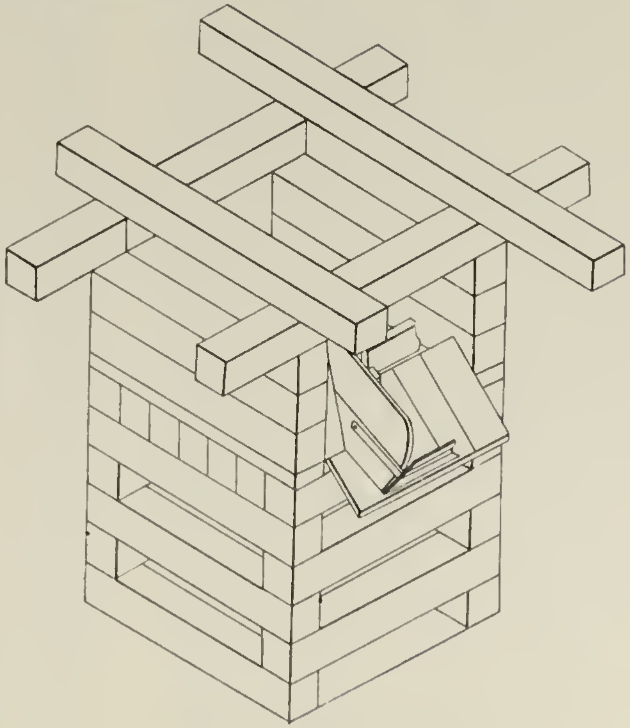


Figure 19.- Standard waste chute

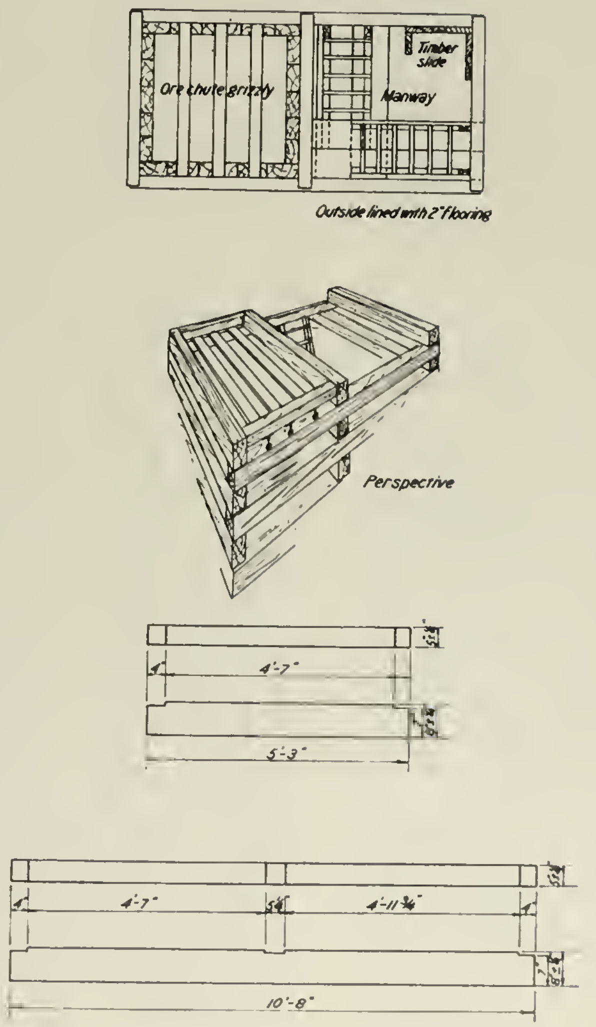


Figure 20.- Detail of framing, standard 2 compartment cribbed chute

The method employed in laying out and preparing a block of ore for stoping is, first, to drive the contact drift and determine by diamond drilling the approximate ore boundaries. After this is done, the vertical pillar scheme and the size of the stopes is determined. The next step is to drive a raise, which will later serve as a waste raise, through to the next level. This is a standard 6 by 11 foot raise, with a cribbed manway compartment, the chute side being left untimbered. This raise is usually located as near the center of the stope as possible, as shown in Figure 13. In the smaller stopes only one waste raise is used, but in stopes 100 feet or more in length, two raises are used in order to insure continuous production and to provide better ventilation, the second raise being placed near one end of the stope (see fig. 14).

Where two raises are used, the timber is stripped from both of them, but where only one is used, the timber is left in. This insures a ventilation outlet from the stope through the manway compartment even while the raise is being used for filling. In the case of the two-raise stope, only one of the raises at a time is used for waste fill, the other being left open for ventilation, and they are used alternately to meet the requirements of filling.

After completion of the waste raise, silling operations are started from this raise 21 to 25 feet above the rail. The sill floor is made 7 feet high. At the same time chute raises are started from the drift and driven 1 or 2 feet above the level of the sill, so that as the sill advances the raises are easily found. The chute spacing is 16-1/2 feet center to center in the massive sulphide and 22 feet in the schist and porphyry. The closer spacing is used in the massive sulphide for two reasons: The ore is very heavy and the throwing distance with the shovel must be reduced; also, it breaks into fragments with very sharp edges, causing excessive wear on timbered chutes. Double 10 by 12 inch stringers, upon which to rest chute cribbing or one type of offset chute (figs. 15 and 16) are placed over the chute raises so that the top of the upper stringer is level with the stope flooring (fig. 17 A). On these stringers is placed a grizzly of 60-pound rails, ball down, with 11-inch openings. This grizzly is for use in mining the second cut, and is removed before raising the chute and replaced for each successive cut. Figure 18 shows the method of timbering a single chute, and the spacing blocks for grizzly rails. Figure 20 illustrates the standard two-compartment cribbed chute.

After the sill has reached the pillar lines and the ore limits, a second 7-foot cut is started from the waste raise. In stopes having two waste raises, the cut is started from the raise near the end of the stope. All drilling is done with a drifter, using two rows of flat holes for the 7-foot cut. As many holes as can be used effectively are drilled from each set-up by fanning them out. These holes average about 7 or 8 feet in depth. They are loaded with 50 per cent strength gelatin dynamite, and detonated with No. 8 blasting detonators.

The ore, particularly the massive sulphide, breaks with about 30 per cent of the fragments too large to pass through the 11-inch grizzly opening, thus necessitating considerable secondary blasting. Spacing the holes closer in the primary blasting has been tried with the hope that it would give a better fragmentation, but it resulted only in a lot of cut-off holes and much powder left in the muck pile. The secondary blasting is done with 35 per cent strength gelatin dynamite and consumes 20 per cent of the total explosives used.

For each cut a temporary shoveling floor of 2-inch planks 5 feet 4 inches long is laid. This floor is recovered after shoveling is completed. The broken planks, which do not exceed 10 per cent of the total, are used in slabbing the outside of the chutes, and the unbroken boards are used again in the next floor.

After two or three rounds have been blasted on the new cut, shoveling operations are started and follow at from 10 to 25 feet behind the mining to the end of the stope. About 2 per cent of the ore broken is sorted out as waste by the shovelers, who are allowed 90 cents a ton for waste sorted. As soon as the shovelers pass a chute and there is no more ore to go into it, the timbermen raise the chute 7 feet and filling is started. To start filling, enough waste fill is dumped down the waste raise directly into the stope, and leveled off by hand, to form a bench large enough for the erection of a bulkhead waste chute (fig. 19). Filling is done by hand-tramming in specially designed scoop cars (fig. 21) run on sectional track (fig. 22) laid on the fill.

Before introducing the first fill, a sill floor is laid, consisting of 4 by 12 inch sills at 5-foot 4-inch centers, and a double floor of 2-inch planks 10 feet 8 inches long placed so as to break joints.

Along the pillar line a gob fence (fig. 17 B) is built on each floor before the introduction of waste. This fence is built of 6 by 8 inch posts 8 feet long, spaced 3 feet 7 inches center to center, covered with 2-inch planks 7 feet 2 inches long with staggered joints.

The vertical section shown in Figure 11 illustrates the sequence of operations in cut-and-fill mining.

Filling

The material used for filling is the waste rock from stripping operations in the shovel pit and the waste from underground development work. It consists mainly of diorite and porphyry, some of it being quite fresh and consequently coarse, with enough fines to fill all voids. This waste is delivered to the various levels through two main waste raises, one directly from the shovel pit and the other from the Hopewell haulage tunnel. Because of the distance the waste falls in these raises, which are intentionally kept empty at the top, considerable crushing action takes place; the waste as delivered to the stopes is considerably finer than when dumped into the tops of the raises. Only waste which is practically free from sulphur is used, a necessary precaution on account of the great fire hazard in this mine.

From the main waste raises the waste is trammed by the same equipment that hauls the ore, to the tops of the waste raises leading from the stopes, and is spread in the stopes as described above.

No effort is made to tamp the waste. The action of the water from the drills seeping downward and the force of the falling ore from blasting cause a settlement of about 8 inches for each 7-foot layer of fill. Subsequent settling in old stopes is very slight. The result is a well compacted waste fill which offers very little difficulty in later mining operations in the removal of either the level or vertical pillars.

Inclined cut-and-fill stoping has been used in a few special cases where the conditions seemed to warrant the change from the horizontal method, as for example, in the extraction of level pillars where the weight on a flat back and square brow had to be relieved or square-setting resorted to, or in a few isolated ore bodies whose general shape makes inclined work preferable to horizontal.

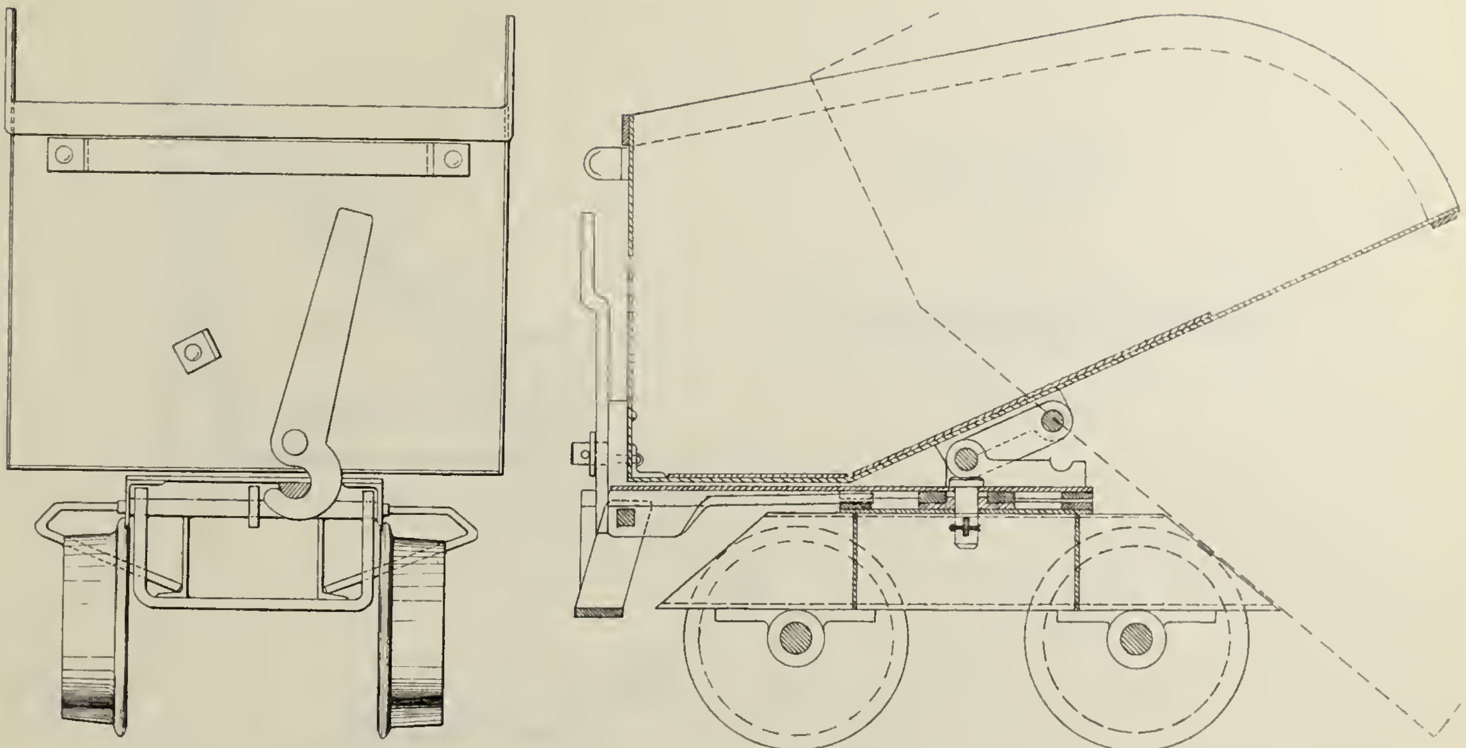


Figure 21.–18 cu. ft. scoop-body car

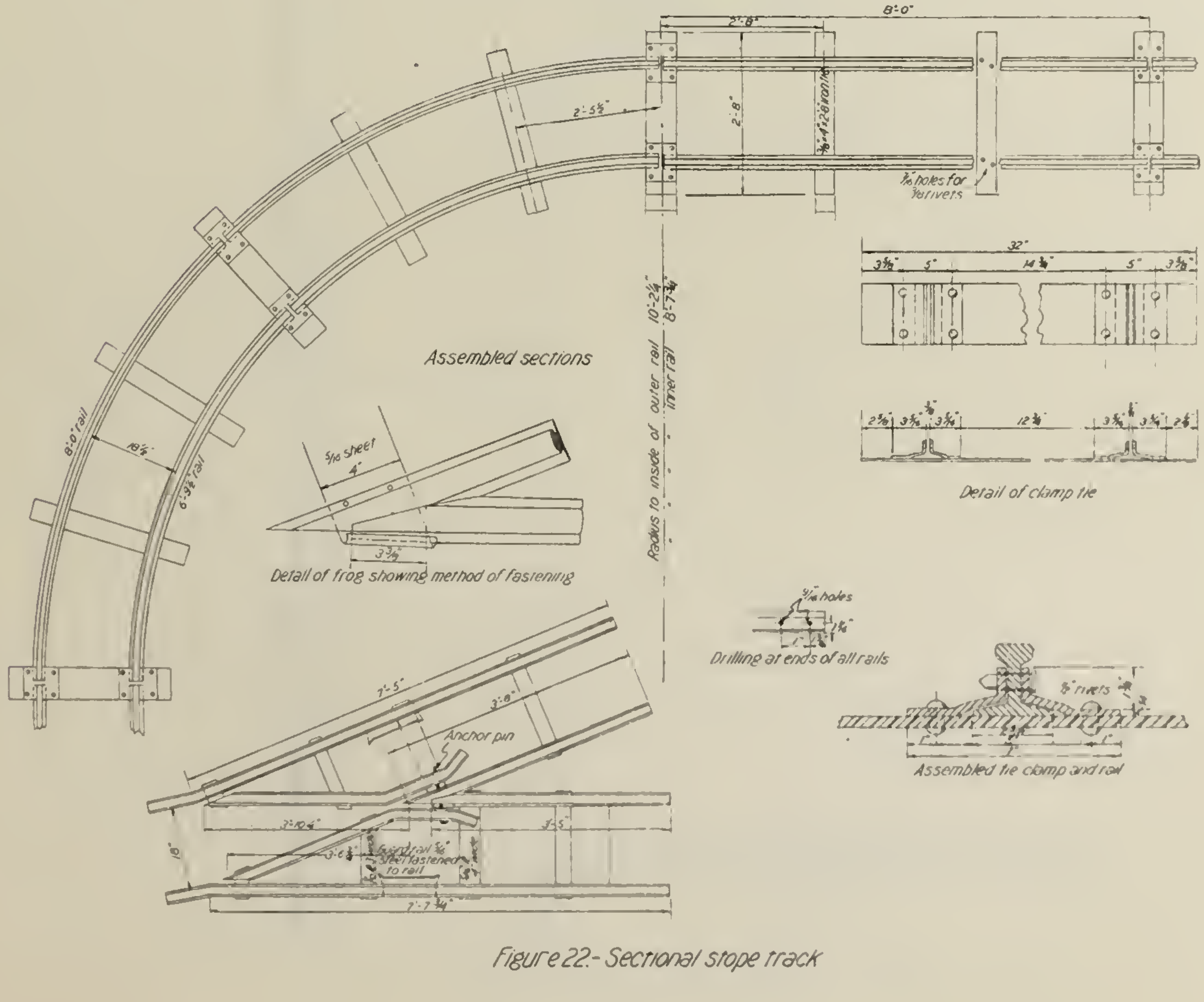


Figure 22.– Sectional stope track

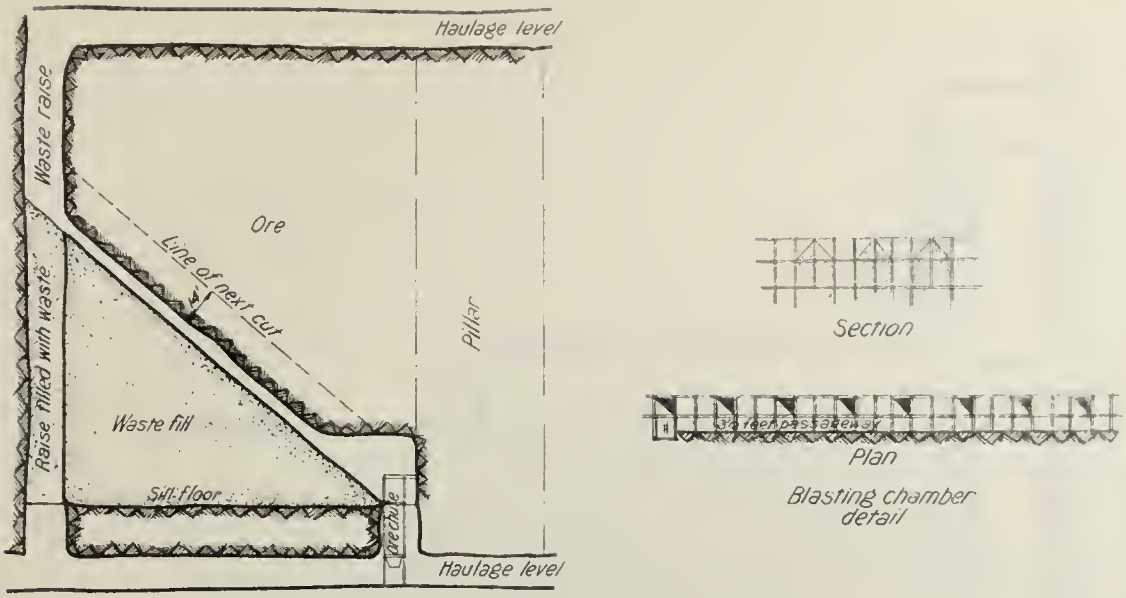


Figure 23.-Typical inclined cut-and-fill stope

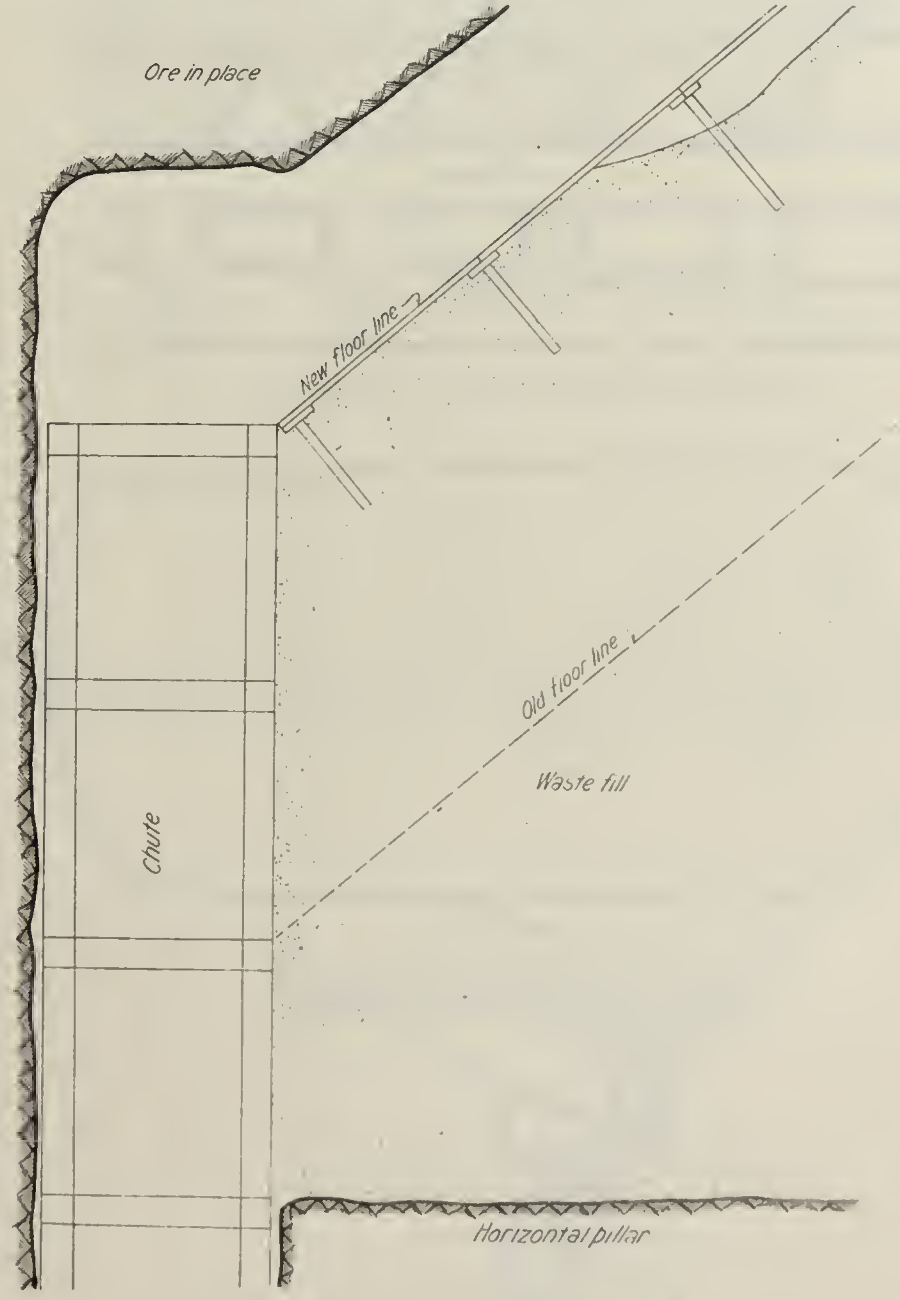


Figure 24.-Detail of flooring, inclined cut-and-fill

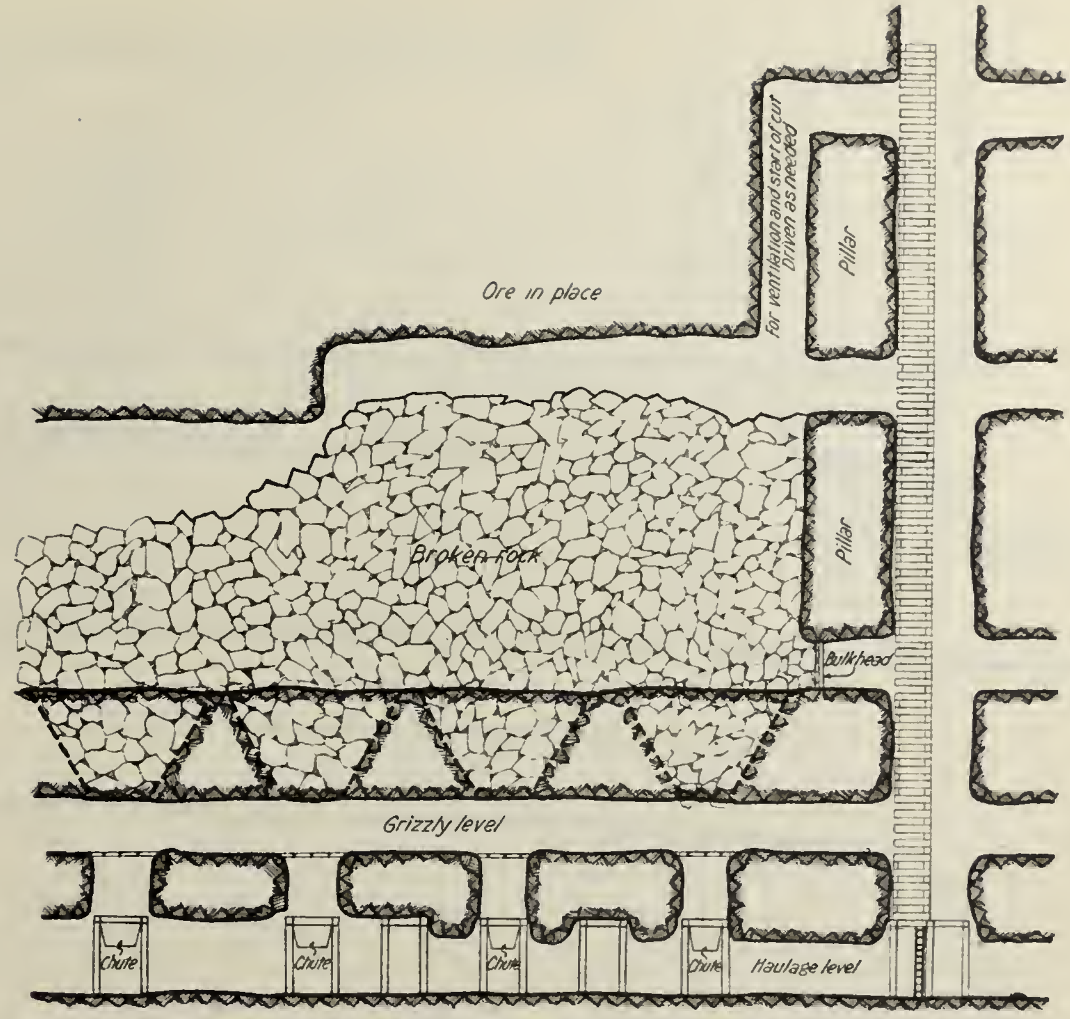


Figure 25.- Shrinkage stope, longitudinal vertical section

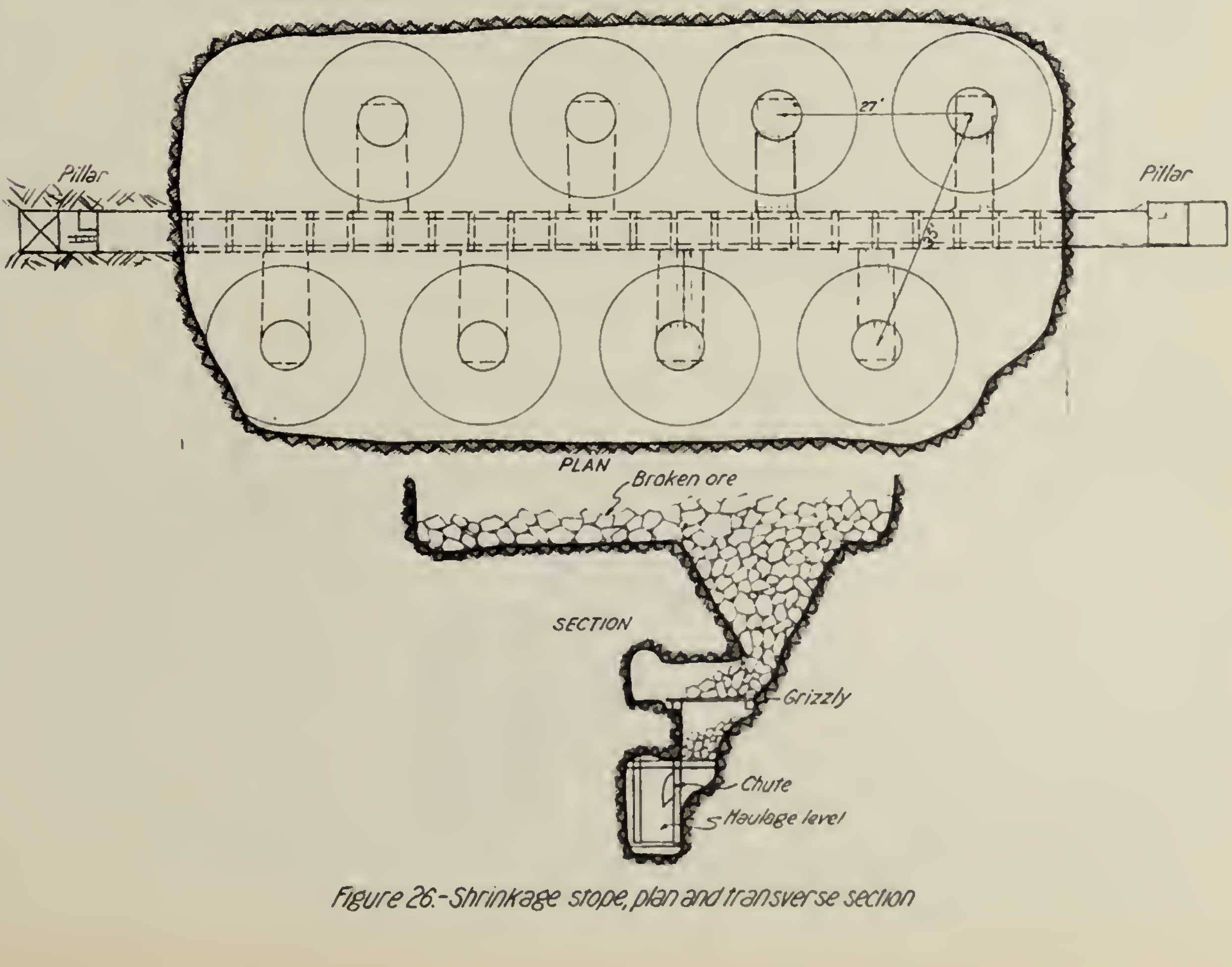


Figure 26.- Shrinkage stope, plan and transverse section

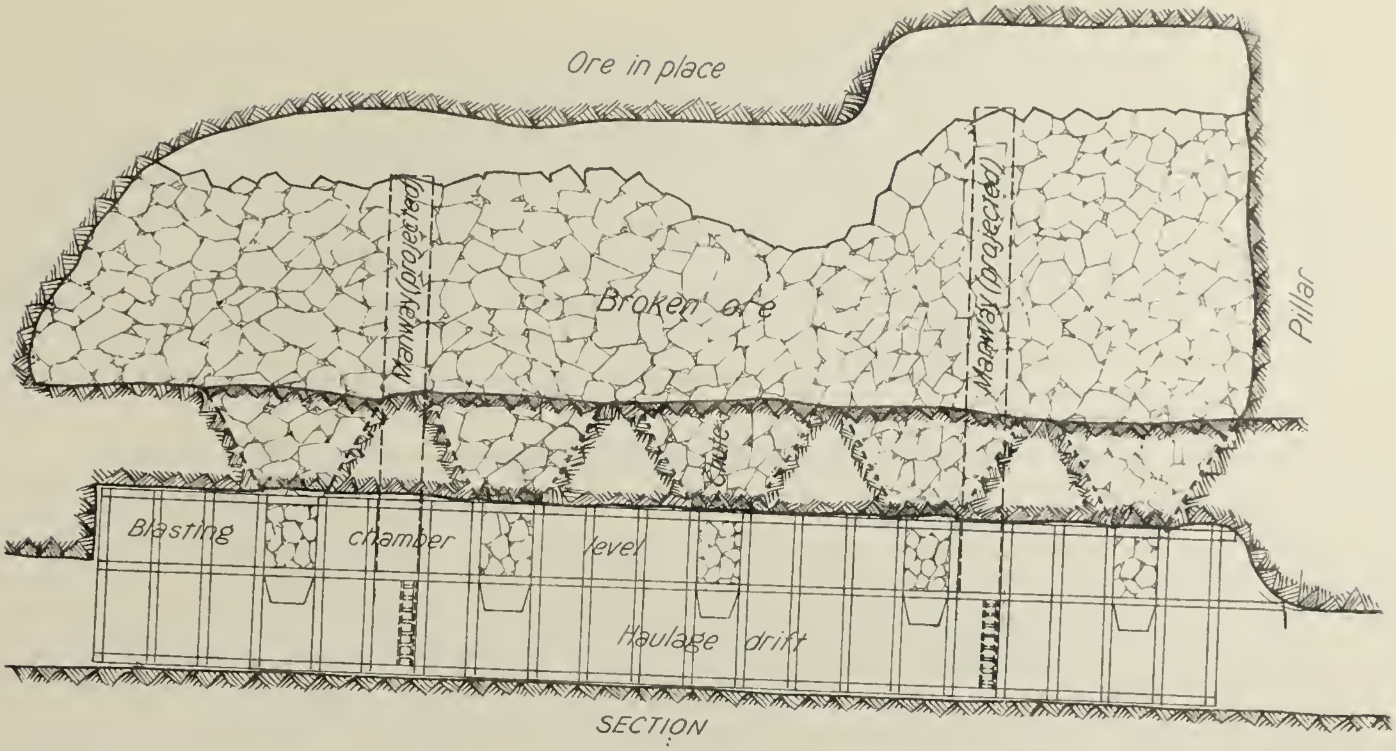


Figure 27.-Shrinkage stope, longitudinal vertical section

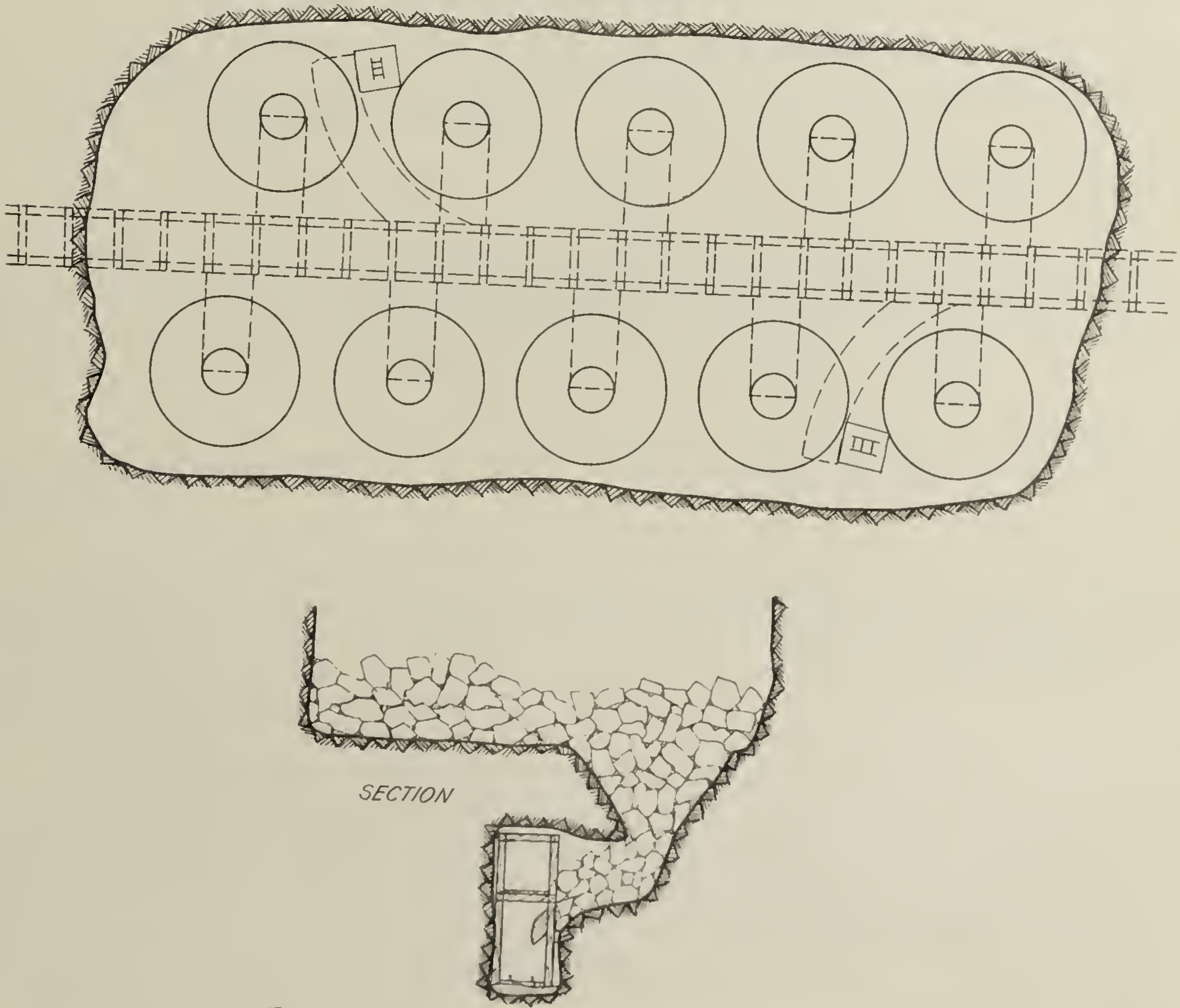


Figure 28.-Shrinkage stope, plan and transverse section

In developing an inclined cut-and-fill stope, a waste raise is driven at the end that is to be carried as the top of the incline. Then the whole area to be stoped is silled horizontally. In some cases where the stope is to be of considerable size, chutes are placed throughout the sill as in horizontal cut-and-fill. These chutes, except those at the opposite end from the waste raise, are later bulkheaded and abandoned when the toe of the incline has passed them. After silling operations are complete, two cuts inclined at 40° from the horizontal are taken at the waste raise. This is followed by filling, then another cut is mined and filled, and so on until the incline is fully established and the toe of the fill has reached the chutes at the end of the stope.

When the incline is established, a row of square sets is placed along the wall over the chutes for the protection of the shovelers (see fig. 23). A horizontal cut is next taken from the back, starting at the wall and meeting the incline. The incline is then mined from the bottom up, usually beginning in the center and carrying the bench up as an inverted V. After the cut is completed and the broken ore removed, the new floor is started at the bottom and placed at an angle about 3° steeper than the natural angle of repose of the waste fill. Only 5 to 10 feet, measuring along the incline, of floor is placed at a time, and waste fill is introduced at the top of the incline. This method insures a compact fill under the floor and reduces the breakage of flooring to less than 10 per cent (see fig. 24). The fill is introduced through a raise at the top and trammed along the crest in scoop cars.

This method is considerably cheaper than the horizontal cut-and-fill, but on account of difficulties in sorting waste and placing bulkheads under slabs in the back it is not applicable to all ground in this mine.

Shrinkage

Shrinkage stoping is used only to a very limited extent in the main ore body because of the necessity of subsequently building gob fences and filling with waste with an attendant accident risk. Consequently, shrinkage is now used only in isolated ore bodies in the massive sulphide where the standing quality of the ground is excellent.

Although there are two methods of developing shrinkage stopes, the operation of the stope in either case is the same. One method is shown in Figures 25 and 26 and the other in Figures 27 and 28.

In either method, the stope is silled in the same manner as a cut-and-fill stope, and successive 7-foot cuts are mined, starting at the raise, and using flat holes. The small cut is used to avoid burying large bowlders so deeply that they can not be drilled and blasted before being drawn down. Just enough ore is drawn to allow comfortable working room between the broken ore and the back.

After completion of breaking, the back of the stope is carefully barred down and in some cases gunited to insure a safe back until the stope can be drawn empty and filled. During the drawing-off operations, men are kept in the stope to bar down the walls and leave them in safe condition. After the stope is drawn empty the chutes in the bottom are bulkheaded, and the stope is floored in the same manner as cut-and-fill stopes. Filling then begins by dumping directly from the level above. As the filling progresses gob fences are built against any ore walls that may be in the stope.

This method is approximately 10 cents per ton cheaper than cut-and-fill stoping,

but on account of the high accident risk, particularly during the filling period, it is not used except where conditions are ideal.

Horizontal Square-Set

Square-setting is used in the extraction of the vertical pillars between cut-and-fill stopes, and also in some areas in the main stopes where the ground is heavy. The size of square-set stopes varies considerably. In the vertical pillars the sections are necessarily small, rarely exceeding 35 sets in area, while in the main stopes there may be as many as 250 sets on a floor. The sets are 5 feet 6 inches square by 7 feet 2 inches high. The timber used is all 10 by 10 inch native pine. The shoveling floor is made of 2-inch plank, and the working floor of 3-inch plank held on by 2 by 2 inch lagging strips nailed to the caps and girts.

The operation of the stope follows the usual square-set and fill practice (see fig. 29). Never more than two floors are opened at a time, and the fill is kept as close behind the muckers as possible. The fill is introduced through a raise to the level above, and distributed in the stope by means of narrow end-dump cars running on sectional track with short-radius curves so as to obviate the necessity of turntables or turnsheets in the stope. The narrow car and special track allow plenty of clearance between the car and the posts.

An adaptation to square-set practice of the chute shown in Figure 30 is used to reduce the number of chutes and to shorten the shoveling distance.

Figure 31 shows the method of offsetting a main chute when it becomes necessary due to an offset in the ore limits.

Inclined Square-Set

As shown in Figure 32, the inclined square-set method is used principally in the extraction of level pillars. Its main advantages are an increased tonnage from a given area, and a lower cost per ton as compared with horizontal square-setting. It also has the advantage that there is less weight in the brow, since there is no large square brow at any place. The only serious disadvantage is the difficulty in sorting.

Such a stope is started from a raise in the center of the stope, usually by taking out four rows of sets on the second floor and two on the third before any fill is introduced. After the incline is established, stoping on each new cut is started at the bottom and progresses upward to the crest. One side of the rill is mined while the other side is being filled. The mining procedure is similar to that for any other square-set stope; the only additional precaution necessary is that more care must be exercised in the side blocking to prevent the side thrust of the waste, which is confined to a steeper angle than its natural angle of repose, from causing a side movement, and distortion of the sets. The greatest contributing factor to the success of this stoping method has been the way in which the floor has been placed and the waste introduced. This is shown in Figure 33. The floor is placed at a slightly steeper angle than the angle of repose of the waste, so that when the waste is dumped in at the crest, little or no tamping of waste under the floor is necessary. There are a few voids occasionally under the caps which have to be hand-filled.

Top Slicing

The top-slicing method has been used only in a special case in the extraction of

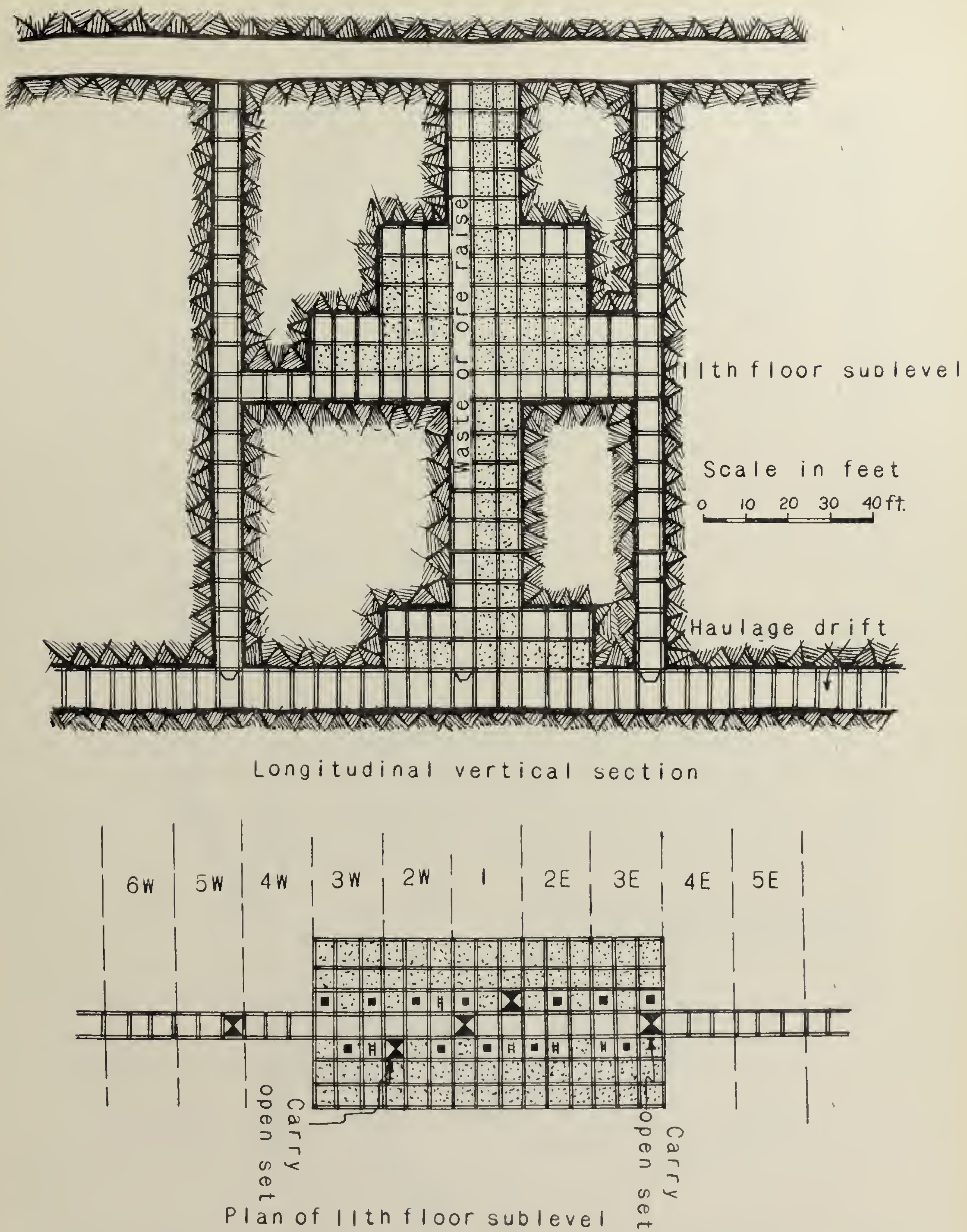


Figure 29.- Square-set method of mining vertical pillar

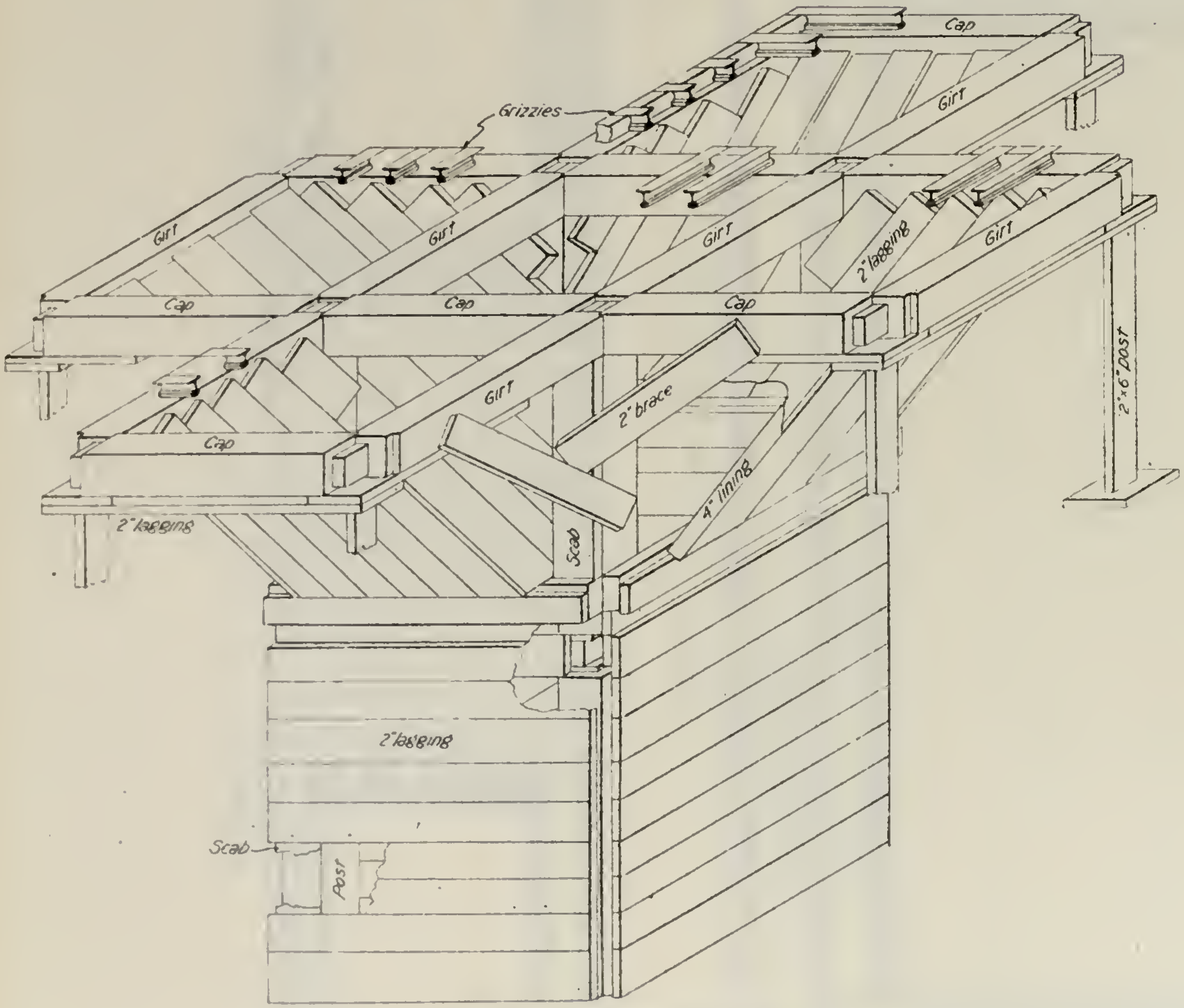


Figure 30.- "Cloverleaf" offset chute

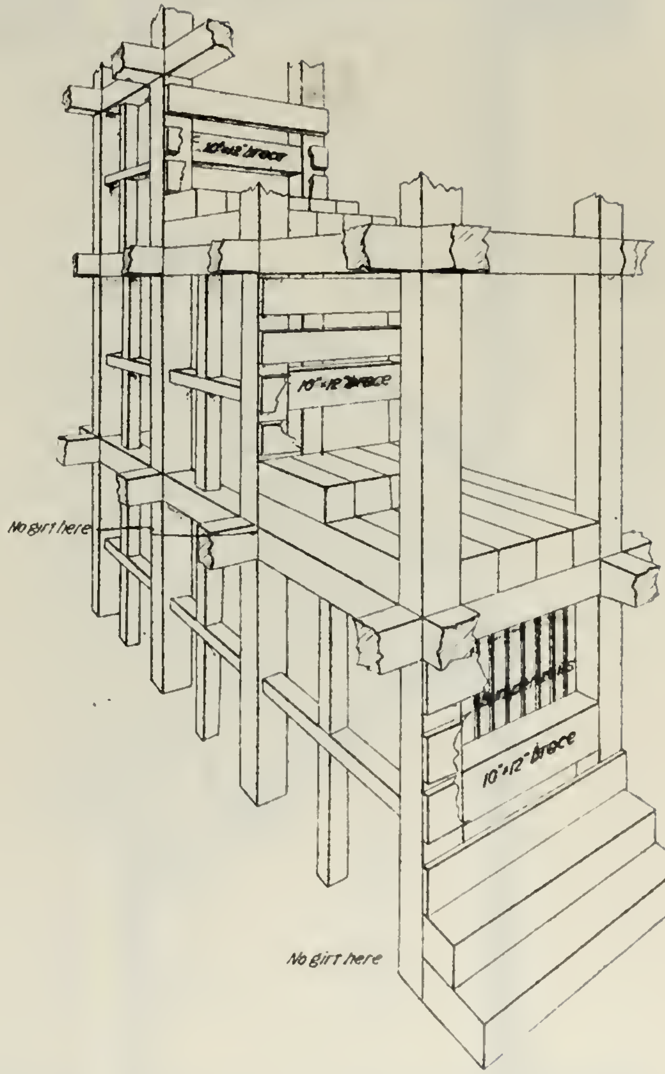


Figure 31.- Offset chute, square-set stope

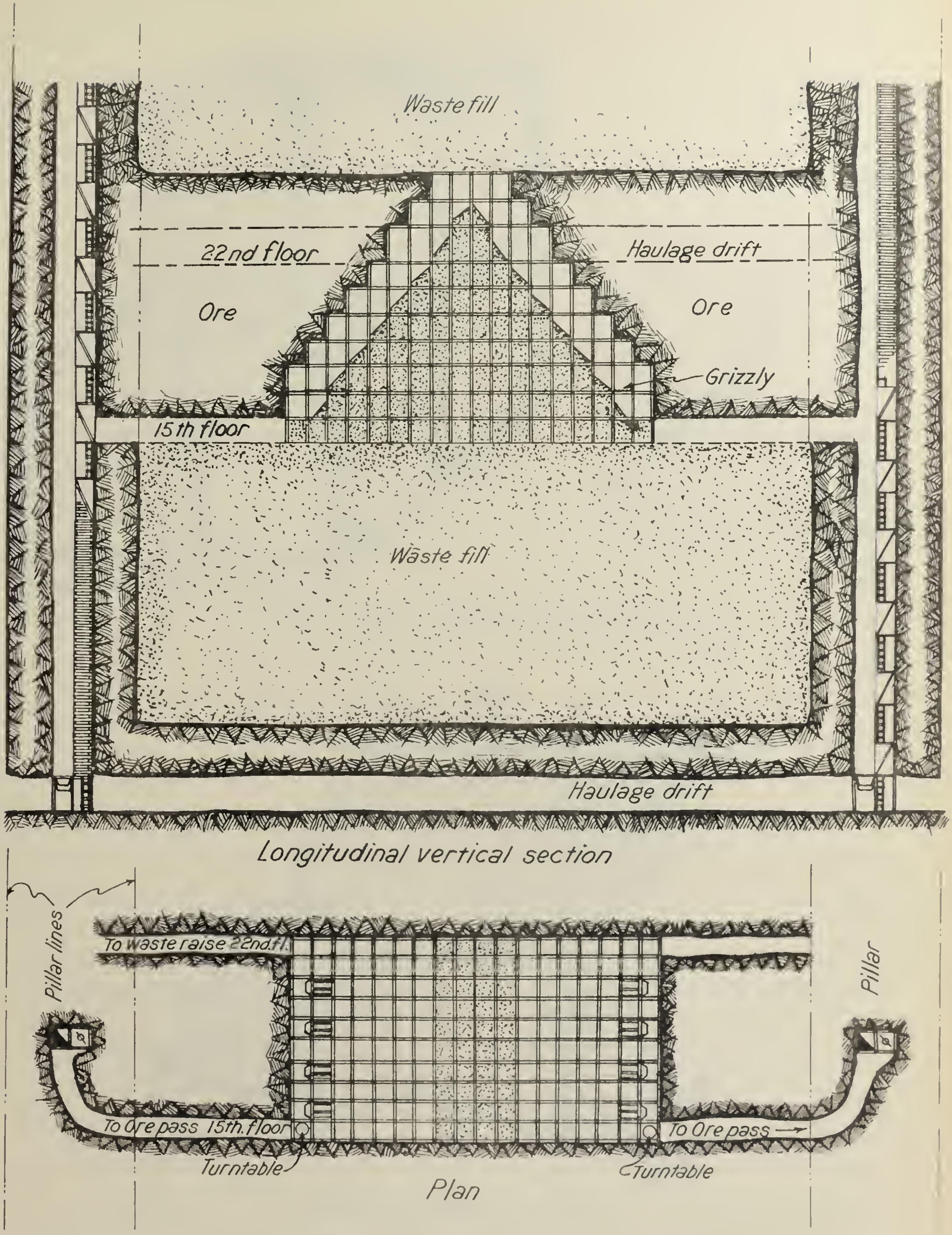
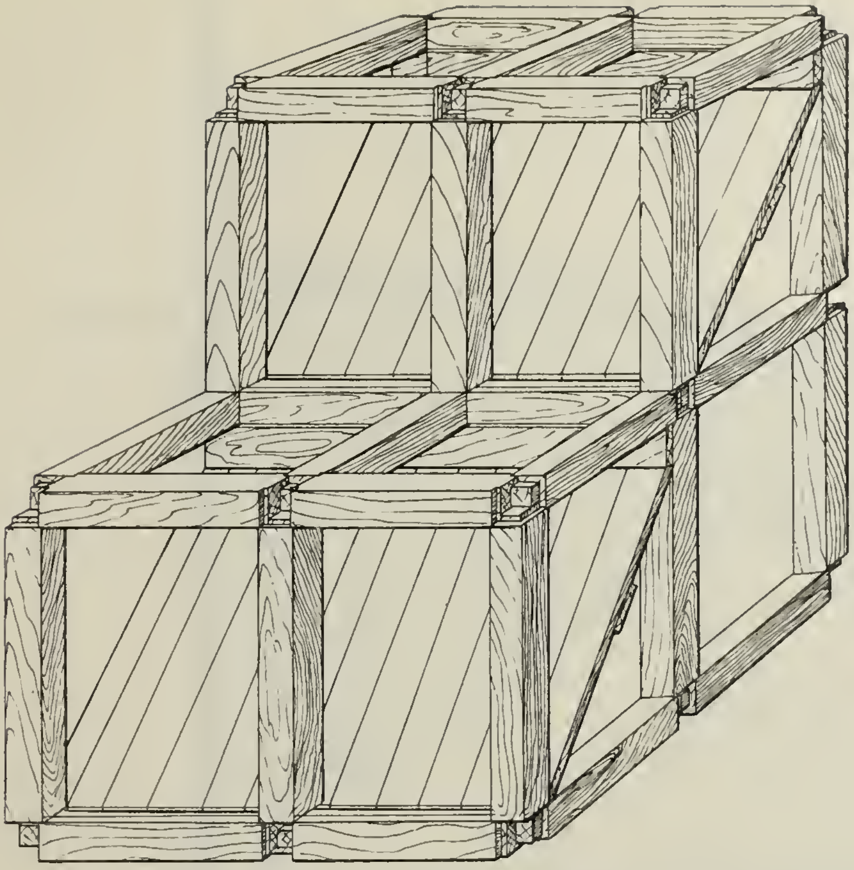
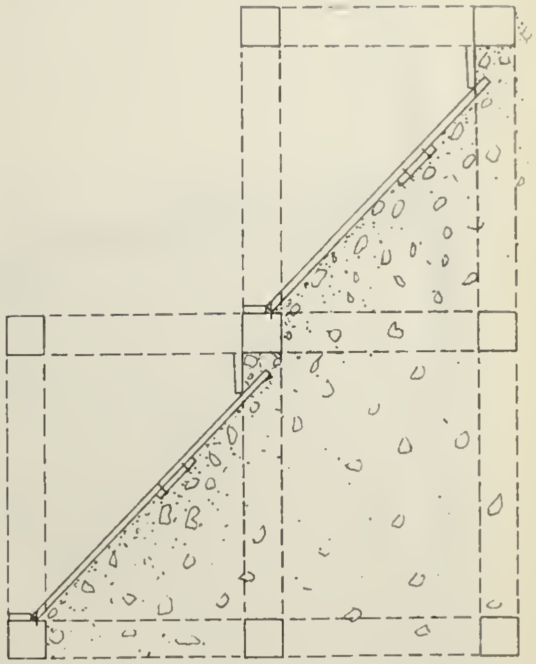


Figure 32.- Inclined square-set method of mining horizontal pillar





Section through center of set showing waste fill

Figure 33.- Method of flooring in inclined square-setting

Figure 34.- Plan of No.2 top-slice stope

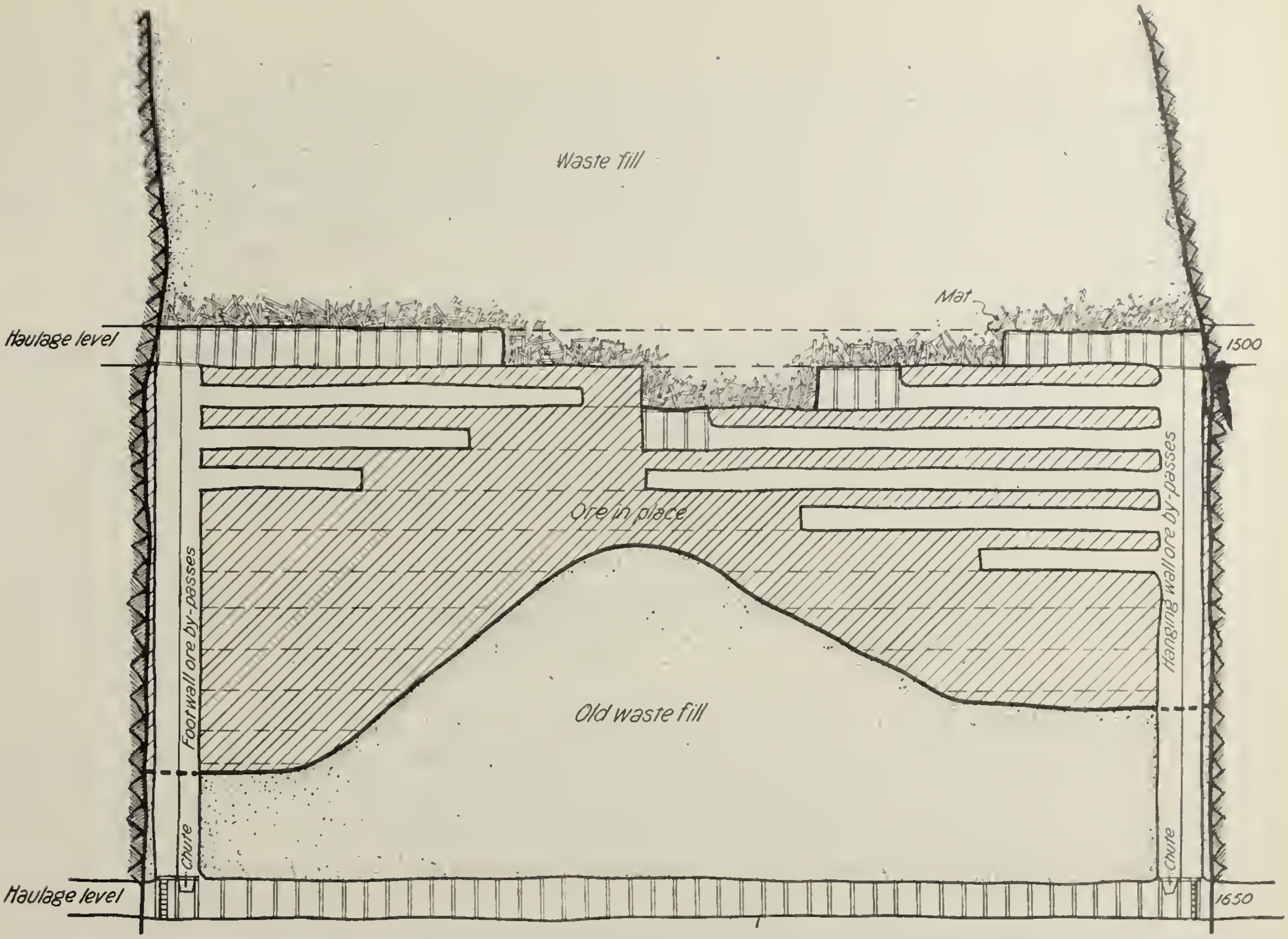


Figure 35.-Vertical section through No. 2 top slice stope, section C-D

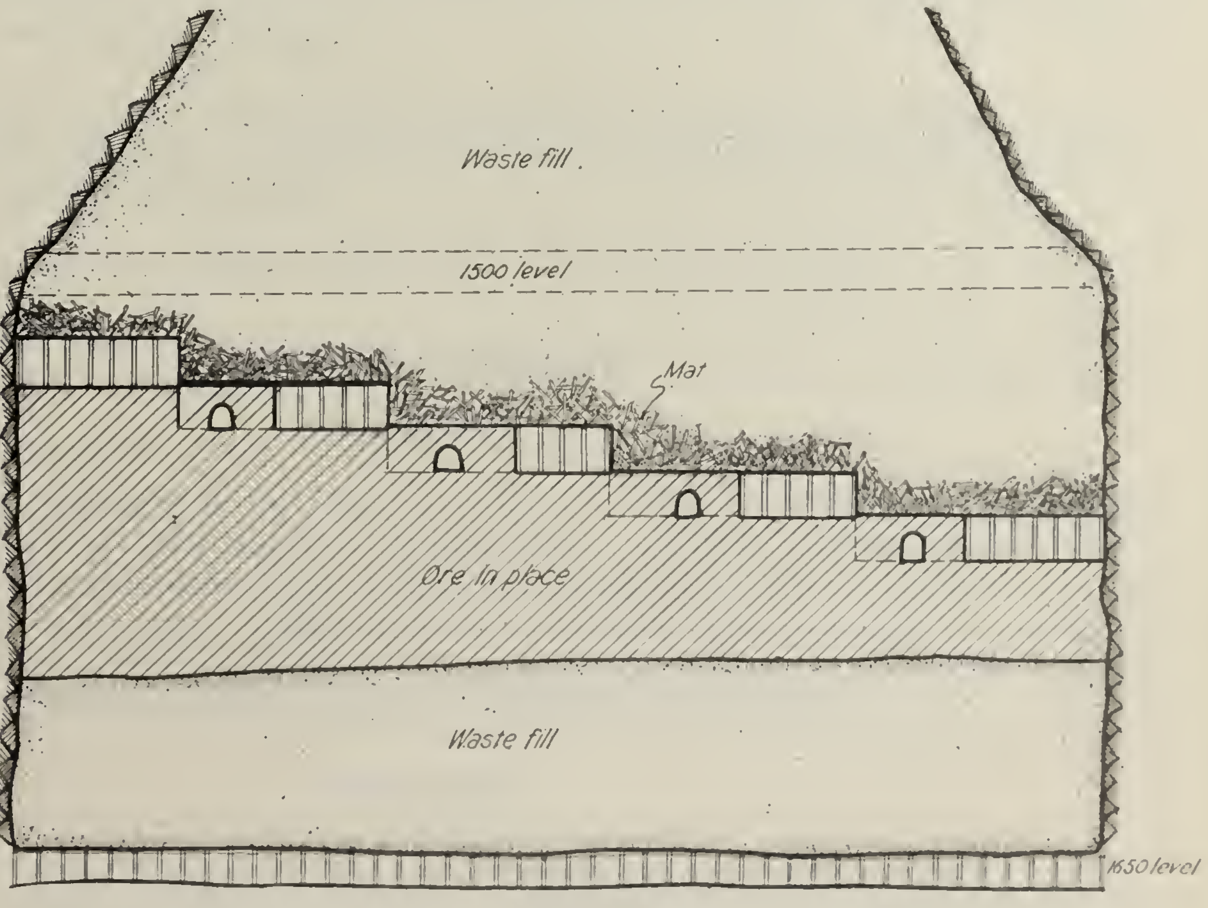


Figure 36.-Vertical section through No 2 top slice stope, section A-B

a caved level pillar over an old cut-and-fill stope of considerable area. The method is not applicable to the usual conditions at the United Verde on account of the high sulphur content of the ore and its accompanying fire hazard. In the instance noted it was necessary to slice under a diorite fill with very little binding material in it, and at the same time to prevent caving of the hanging wall. For this reason the old stopes above were kept fully filled as slicing progressed, which effectively prevented any major caving of the hanging wall.

Because of the caving of so large an area, all chutes in that area were crushed and entirely lost. This made it necessary to run raises near each wall and drive panel drifts from them to the center of the stope, where actual stoping was started. The stope panels are mined by retreating from the center to the raises on each wall, the ore being shoveled into scoop cars and trammed to the raises. Never more than 25 feet in length of panel is open at a time.

The panel drifts are driven untimbered about 4 by 6 feet in cross section and 35 feet center to center. The slices are 11 feet high.

A very tight floor is laid by placing 4 by 12 inch sills at 5-foot 4-inch centers at right angles to the panel drifts and laying a double 2-inch floor on top of them. The posts are 10 to 14 inch native pine unpeeled poles, and are placed at approximately 5-foot 4-inch centers with 6 by 12 inch head-blocks, and are so placed under the sills of the floor above.

Both primary and secondary blasting of the ore are done with gelatin dynamite and gutta-percha fuse, but because of the fire hazard the slices are shot down with a special permissible powder and electric detonators.

This method is cheaper than square-setting, and the tonnage it is possible to remove in a given time is much greater. The latter factor was the controlling one in the adoption of top slicing for this particular stope.

Details of the stope layout are shown in Figures 34, 35, and 36.

Underground Transportation

Hand Tramming

Hand-tramming is used on the levels only where the hauls are short and the tonnage insufficient to keep a motor crew busy. It is also used in sublevel gangways where a transfer of stope ore is more economical than would be the raises or drifts necessary to develop irregular offsets in the ore body. Figure 21 shows an 18 cubic foot scoop-type car developed for this purpose. It is used with 16-pound, 18-inch gage sectional track on steel ties as shown in Figure 22. This car is also used for spreading waste in stopes.

Motor Haulage

Practically all tonnage is handled from the stopes to No. 5 shaft by motor haulage. The length of haul averages 1,000 feet. Trolley locomotives have been largely replaced by 4 and 6 ton storage-battery locomotives, which handle sixteen to eighteen 30-cubic foot rocker-bottom cars. These cars weigh 2,500 pounds, and are equipped with 12-inch roller-

bearing wheels. A particular feature of this car is a double trunnion, which gives a minimum overhang in the dumping position and permits dumping in the standard width gangways when handling waste for stope fill. The car is dumped by means of a piece of drill steel, one end of which is inserted in a shallow hole in the drift wall, the other end being placed against a lug on the side of the car. As the car moves forward, the drill steel forces the car over. Several mechanical dumping devices have been tried, but the drill-steel method is simple and effective.

Each level is equipped with a charging station with steel platforms on either side of the track. The tops of these platforms are the same height as the frame of the motor, which permits the battery boxes to be rolled on or off the motors as required. Each motor has an extra set of batteries which is on charge while the other is in service. The charging sets are of the constant-potential type and operate from 250-volt direct-current circuit.

Figure 37 shows a standard chute pocket for ore-passes from stopes. The steel gate is operated by hand, using an 18-inch forged handle with a slot to slip over the truss strap on the gate.

Track

Forty-pound rail laid at 18-inch gage on 6 by 6 by 30 inch ties, spaced at 24-inch centers, is standard for all motor-haulage drifts. Sixteen-pound track, either sectional or laid on 4 by 6 inch ties, is used for hand-tramming. Nos. 4 and 5 manganese steel frogs are standard equipment, and are rebuilt by electric welding when worn down. Switch throws, points, and the necessary rail clamps, spacers, etc., are constructed in the mine shops. A minimum radius of 50 feet is maintained for all motor-haulage track.

Hopewell Tunnel Transportation

All ores from the pit and underground operations, as well as excess waste from the pit, are transported from the main loading bins on the 1000 level through the Hopewell Tunnel to the outside storage bins, which have a capacity of 10,000 tons. Because of the character of the heavy sulphides, particularly the fine calcined material from the fire zone, it is essential that the chute openings be as large as possible to prevent the fines from building up in the bins. Figure 39 shows the general plan of these loading bins, which are very rugged in construction. The gates are operated by air cylinders.

The ore is handled by 40-ton standard-gage cars, provided with compressed-air cylinders for operating the dumping mechanism. Twenty-five-ton trolley locomotives, equipped with two 75-horsepower commutating pole type motors and automatic air brakes, handle a 9-car train at a speed of 7 miles per hour. The average gross weight of the train is 540 tons.

With four motor crews of three men each, 5,000 tons of ore daily are transported over this 1.8-mile haul. The track is standard gage, with 75-pound rail, and has a grade of 0.5 per cent in favor of the load.

Hoisting Practice

Because of the abrasive effect of the hard sulphide ore, all chute, grizzly, and pocket construction is very heavy. Loading pockets (fig. 30), are of 500-ton capacity, and are arranged according to general practice in the Southwest. The upper gate is of the undercut plate type, while the measuring chute has an undercut arc gate with a movable baffle

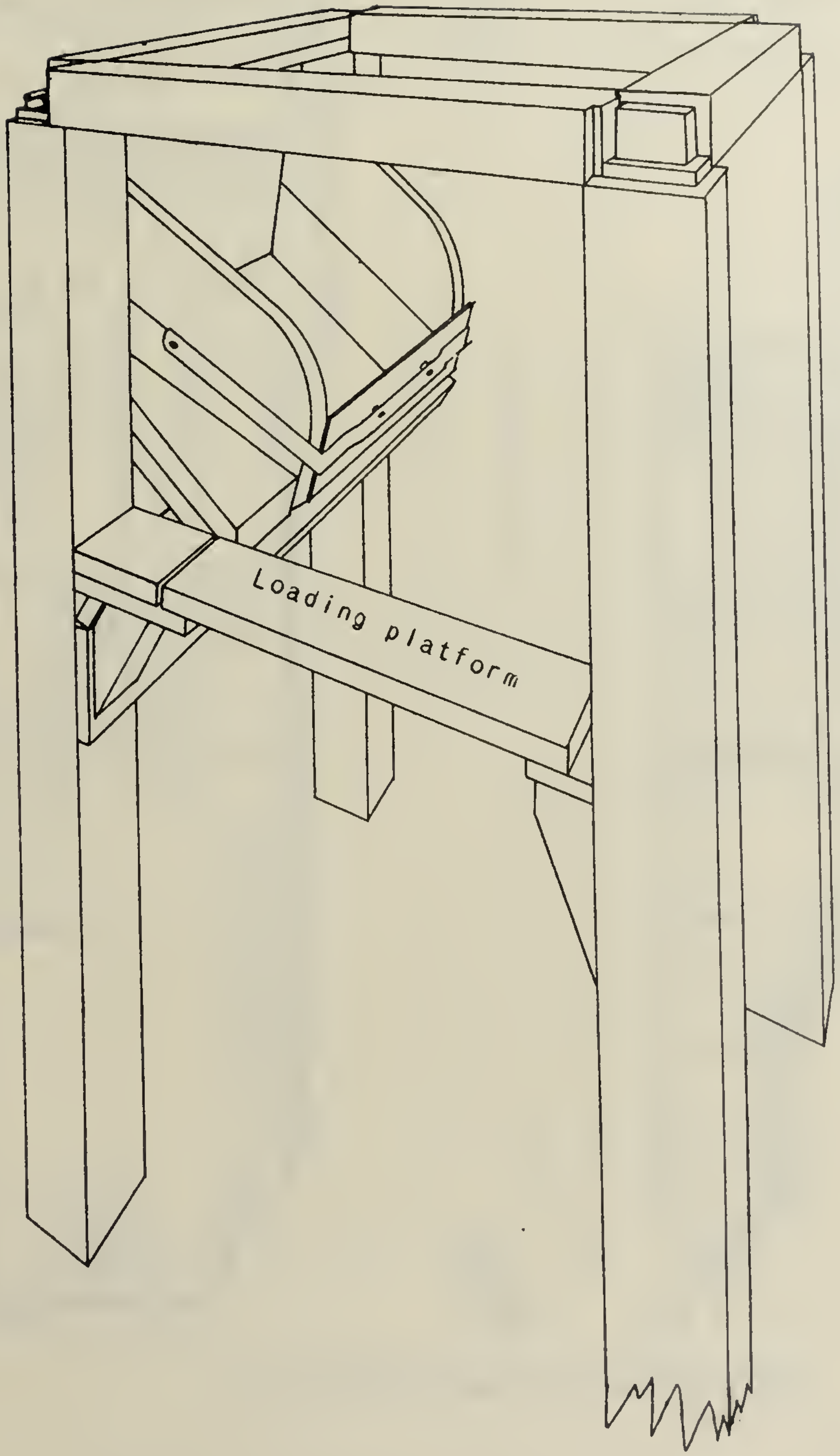


Figure 37.- Standard loading chute

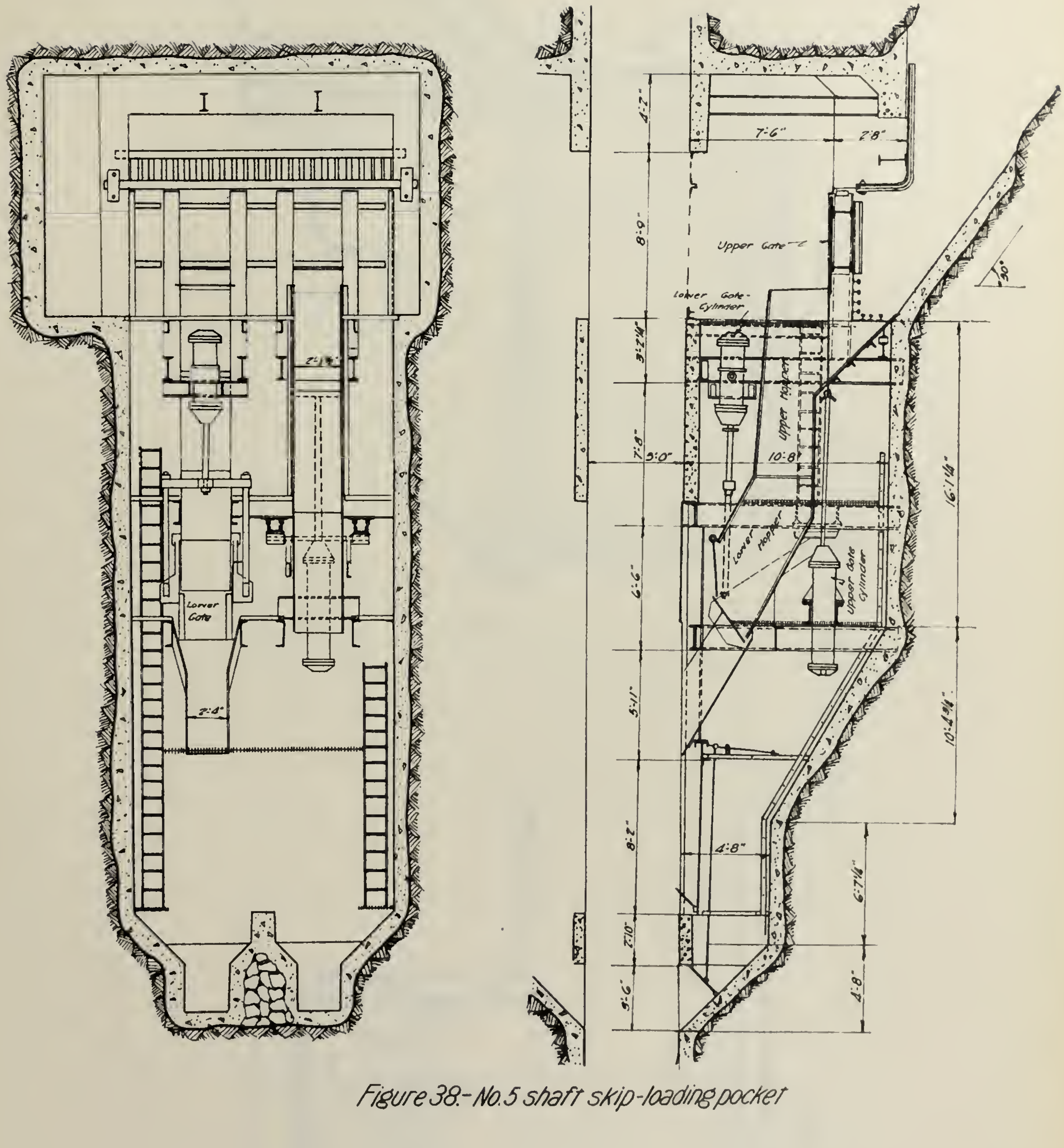


Figure 38.- No. 5 shaft skip-loading pocket

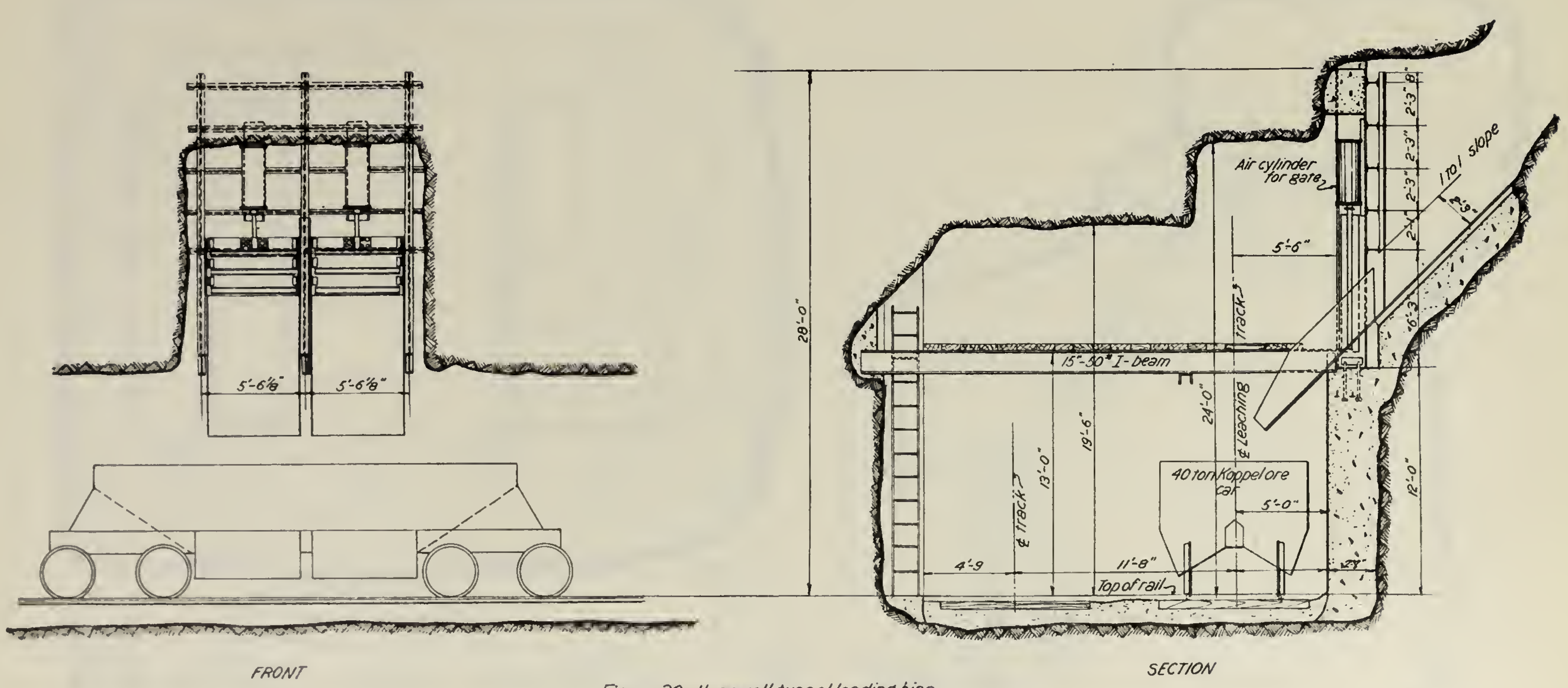


Figure 39.- Hopewell tunnel loading bins

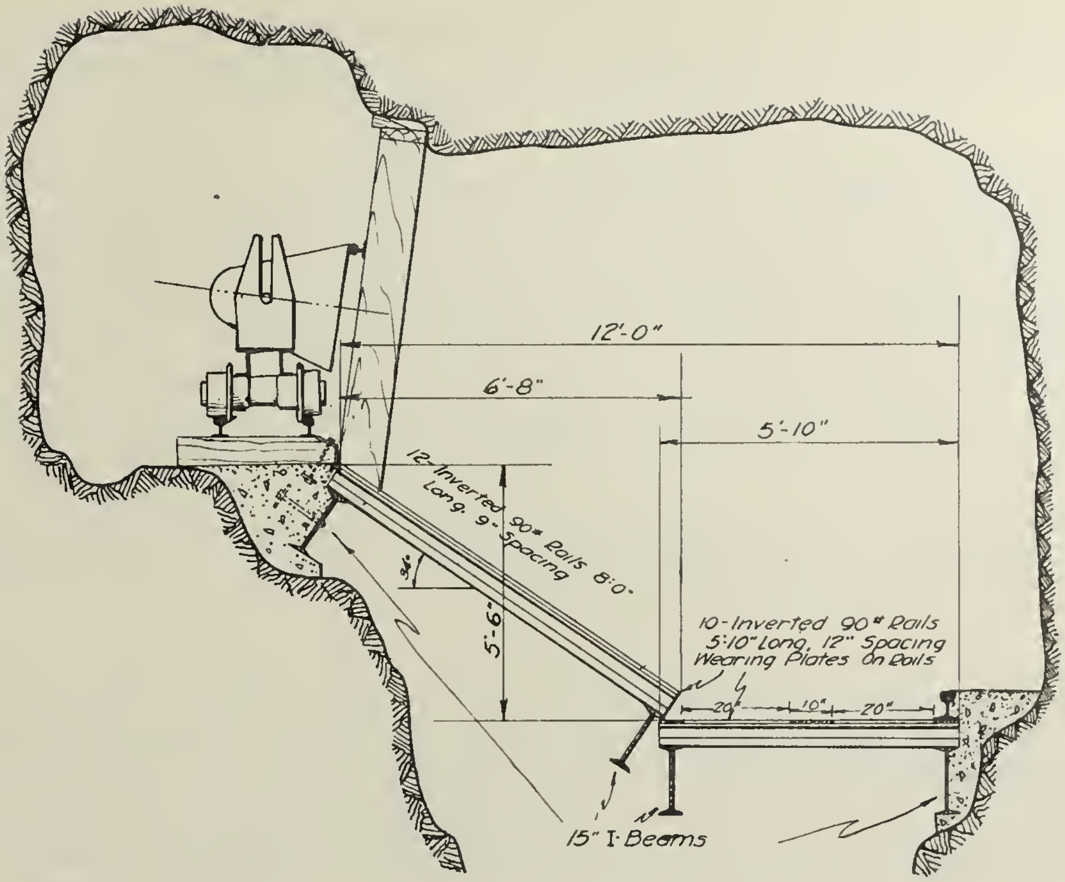


Figure 40.- Standard ore-pocket grizzly

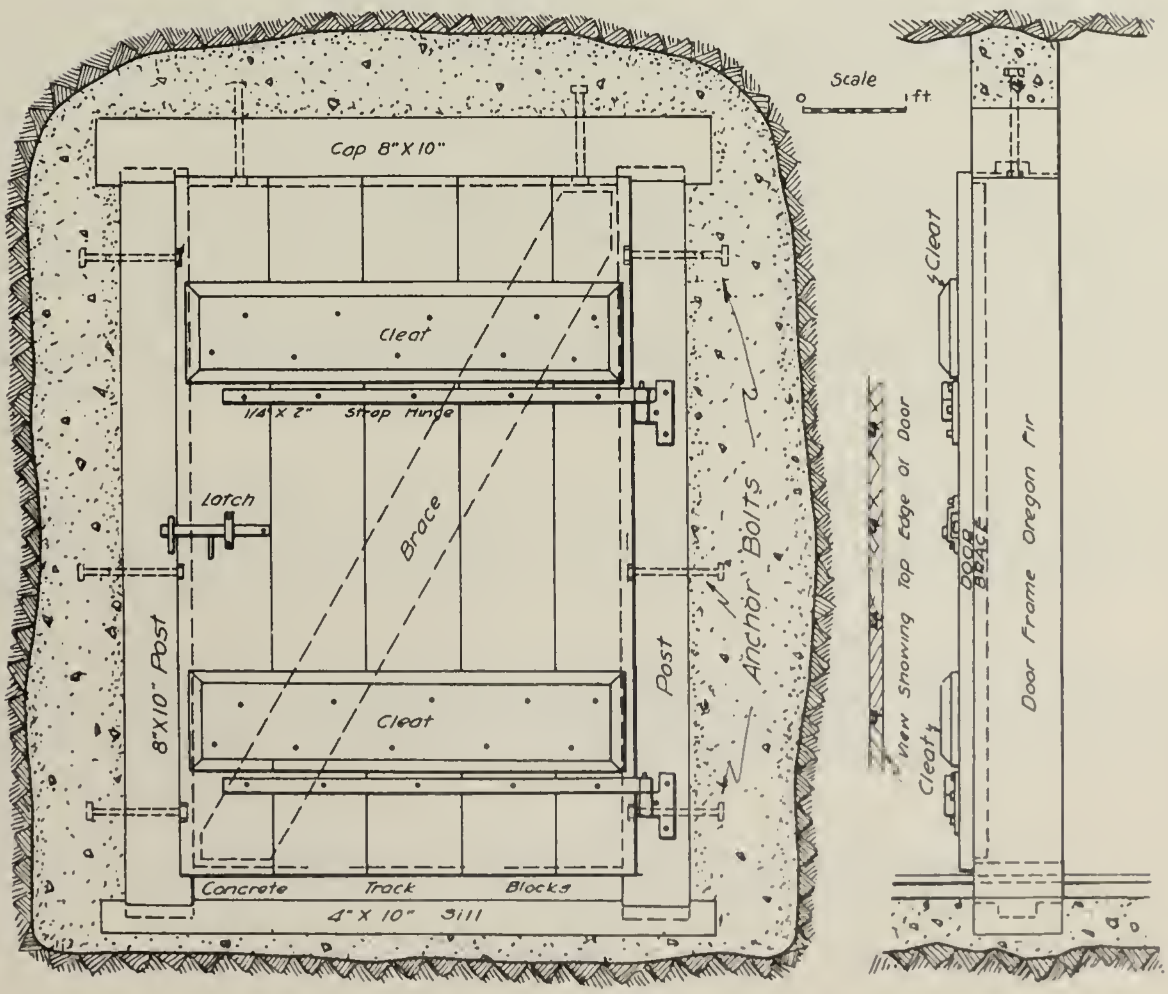


Figure 41.- Standard mine motor door

front. Both gates are operated by compressed air. The average loading time is 7 seconds. The valve on the air line leading into the pocket is provided with a padlock, so that no one but the skip loaders can operate the gates.

Figure 40 shows the grizzlies above each pocket. The grizzly bars are inverted 90-pound rails which have been annealed and faced with a 1 by 5 inch wearing plate. The inclined rails are spaced to give 9-inch openings, and the lower horizontal rails have 12 by 20 inch openings. This construction permits an entire train to be dumped without stopping to bulldoze the bowlders. Each level is equipped with two sets of grizzlies, with transfer raises to the level or loading pocket below. This is essential, as mill ore consisting of schist and porphyry, and direct smelting ore consisting of massive sulphide and high-grade-schist (over 7.0 per cent copper), are handled separately.

The spill from hoisting 2,000 tons daily averages 6 tons. This spill being abrasive, wears the shaft walls, and it has been necessary to build up the curtain walls by means of gunite.

A heavy hinged bulkhead, or deflector, constructed in the hoisting compartment, deflects the greater portion of the spill into the ore pockets. This deflector, located at the 1950 level, is electrically operated by remote control from the engineer's platform, and pilot lights indicate the open and closed positions. The portion of the spill which reaches the bottom of the shaft is drawn through standard chutes into an 18 cubic foot car and hoisted through the manway compartment to the level above.

No. 5 hoist station is located on the 1000 level. It is 47 by 81 feet in plan and 22 feet in height, and is lined with reinforced concrete. The cableway extends at an angle of 60° to the headframe on the 800 level. The sheaves are 10 feet in diameter, and are of the plate-type construction with case-steel liners.

The ore hoist is of the double-drum, single-reduction, geared type, driven by a 650-horsepower, direct-current, 500-volt motor. Power is supplied by a 600-kilowatt generator set with a 20-ton flywheel. Each drum is 10 feet in diameter with a 5-foot face, and holds 2,500 feet of 1-3/8-inch, 6 by 19 hoisting rope in two layers. The hoisting capacity is 260 tons per hour at a rope speed of 890 feet per minute. The total load on the cable is 15 tons. The average life of a cable is two years and the ore or waste hoisted per rope is 760,000 tons.

The skips, which operate in balance, are of 112 cubic foot capacity, equivalent to 8 tons of sulphide ore, or 6 tons of schist ore. They are of rugged construction, have 3/4-inch plate liners, and weigh 7 tons each.

The ore is hoisted to the 800 level, where it is dumped through an electrically operated "selector" to one of three storage bins, depending on the class of ore or waste being hoisted. This selector is operated from the hoistman's platform, and colored pilot lights indicate its position.

The hoisting crew consists of a hoistman, skip loader, and helper.

BONUS SYSTEM

A fair amount of work done safely by an average man in a given unit of time is the

standard for any particular job. The efficiency of the job is calculated as follows:

If a standard for a job is one unit per shift, and a man accomplished one and one-half units per shift, he is rated as being 150 per cent on that job. He receives as bonus one-half of his increase in efficiency over 100 per cent, or 25 per cent of his day's pay rate. The company benefits by the other half. Putting this into a formula, it may be expressed as follows:

$$T_2 = P/S;\ E = T_2/T;\ B = 1/2(E - 100)\ W$$

S = Standard (unit of work per unit of time)
P = Work accomplished
T = Time required for doing P
T = Time allowed for doing P according to S
E = Efficiency in per cent
W = Wage rate per unit of time
B = Bonus paid

Obviously, the major problem is to determine correct standards of work. The setting of standards depends largely upon past performance and upon the judgment of the man setting the standard. The men chosen to set the rates must have had considerable experience themselves in doing and in bossing the work they are rating. Such a man inspires the respect and confidence of the workers. The foremen are consulted whenever a new standard is to be set.

Since 1924, a good deal of work in the mine has been paid for on a contract system. practically all of the new timber work, drifting, crosscutting, and raising is done on contract. However, what we call a contract is not a contract in the true sense of the word, because the agreement is verbal and the man assumes no obligation. The company agrees to pay a certain price for a certain unit of work, but the man is guaranteed day's pay no matter how much he accomplishes. Also, the company retains the right of complete supervision over the work and the manner in which it is done. The contractor has nothing to say about who his coworkers or helpers will be, though, of course, an attempt is made to put men together who have equal ability and who will work together to advantage. Each man is responsible to the company only and they are paid separately by the company. Two examples of contracts follow:

A.

Credit 10 chutes @ $10 =	$100.00	Deduct 5 shifts @ $5.23 =	$26.15
	49.55	5 shifts @ 4.68 =	23.40
Due contract	50.45	Total deductions	$49.55

Due timberman in addition to day's pay: 53 per cent of $50.45 = $26.75
Due helper in addition to day's pay: 47 per cent of $50.45 = $23.70

Division of the profit is made according to the ratio of the day's pay of the two men.

B. A drift is driven under contract at a price of $5 per foot. This price includes mining labor and explosives only, as the mucking is let

separately. The miner drives the drift 100 feet in 20 shifts and uses 30 boxes of powder.

Credit 100 feet @ $5.00	$500.00	Deduct 20 shifts @ $4.95	$99.00
	339.00	30 boxes powder @ $8	240.00
Due contract	$161.00	Total deductions	$339.00

Miner receives $161.00 plus day's pay for 20 shifts.

Two advantages of this type of contract over the bonus system in use here are that the contract is easily understood by the miner and he visualizes his units of work accomplished in dollars, which acts as an incentive to greater industry. Also, in the case of development work, it rewards the miner for saving explosives, while under the bonus system the tendency is to be extravagant and wasteful of explosives.

This type of contract has been very successful in lowering unit costs and in increasing the earnings of the men. It has been tried to a limited extent in mining in stopes and within the next year will be given a thorough tryout for all extraction operations.

A great deal of care must be taken to set a correct contract price in this method of payment because the company does not participate in the bonus, and if the price is set too high the unit cost to the company will be too high.

A great many of our contract prices have been arrived at from the bonus standards, in conjunction with the average bonus earned over a long period of time on a given class of work. For instance, the price of a drift would be set as follows:

The bonus standard for the drift might be 2 feet per shift. The average bonus for drift work was somewhat under 50 per cent, but good miners, under favorable conditions, often made 50 per cent or better. To make 50 per cent bonus in the above drift, the miner would have to average 4 feet advance per day, and this would cost the company $4.95 plus 50 per cent of $4.95, equalling $7.42, or $1.855 per foot for labor. Then, if $1.90 per foot were allowed for explosives, the contract price would be $3.75 per foot. If the miner broke 4 feet per shift and just used the allowed amount of powder, he would make just the same amount of money as if he worked under the bonus system. To make more money on the contract than on bonus, he would have to reduce the amount of powder used per foot or increase the amount of advance per shift. If he fails to make an advance which would have paid 50 per cent bonus under the bonus system, or if he uses an excessive amount of powder, he will not make as much as if he were on bonus instead of contract.

The company gets the work done at a certain fixed price whether the miner makes a good bonus or not, unless he shows a loss. In case of a loss by the contractor, the cost to the company is increased by the amount of the loss over the agreed price.

If the contractor is not doing reasonably well, an investigation is made of the conditions governing the operation, and the contract price is changed if unforeseen difficulties have arisen. If, however, the fault is found to be with the contractor's ability, he is either transferred to some work he is better fitted to accomplish or he is discharged.

The contract price is subject to change at any time if conditions over which the contract has no control are changed. For instance, the formation in a drift may change from soft to hard rock, in which case a new price would be set starting at the contract. If an error, due to misjudgment, has been made in setting the contract price, the change in price is retroactive and starts when the work started if the original price was too low. If the price was set too high, the cut does not take effect until the first of the month following the time the mistake was discovered. Although this gives the men an advantage over the company, it tends to inspire their confidence in the system.

Most of the underground work and all of the mechanical and open-pit work is still under the bonus system. Various types of standards or allowances are set for the different operations according to our ideas of what seems to be the most equitable method of settlement.

Type of work	Unit of standard
Underground work:	
Mining in drifts and crosscuts	Feet per shift advance
Mining in stopes	Cubic feet per shift broken
Laying track, hanging pipe, etc.	Feet per shift installed
Hoisting ore	Skips per shift hoisted
Hauling ore	Cars per shift hauled
Mucking	Cars per shift filled
Open-pit work:	
Trucking ore	Tons hauled per shift
Electric shovel operation	Cubic yards dug per shift
Truck maintenance	Dollars allowance per truck shift
Mechanical work:	
Machine work	Given number of hours allowed per unit of work
Sharpening drill steel	Time allowance per rock-drill shift

The above are only a few examples of the many operations, but they are representative. The drill steel sharpening standard is not based on the number of pieces of steel sharpened because when this method of payment was tried, the operators burned the steel in an effort to increase their production. When burned steel went underground, the bit soon failed and the steel was out again for resharpening. This tended to increase the work in the sharpening shop, and thus the men were paid a premium for doing poor work. Under the present system, when perfect steel is sent underground, a minimum amount of resharpening is required, the shop crew is as small as possible, and a maximum of bonus is made by the men, so that the premium is now paid for good and efficient work. The same method of payment is used in the rock-drill repair shop. They are allowed so much time per drill in operation, so that drills sent down in perfect order will stay down the longest time possible and reduce the work in the shop to a minimum, thus allowing the men a maximum of bonus. This method of reward does very well for maintenance and repair work where certain equipment is in constant use and requires constant attention.

The organization of the bonus department consists of a chief bonus engineer, an assistant chief bonus engineer, an office clerk who posts the time of the workmen on the various job sheets, four bonus engineers who handle the underground work, and two bonus engineers who look after the outside and mechanical work.

The engineers take the cubic feet broken, explosives used, cars trammed, etc., from the shift bosses' distribution sheets each day. They are constantly visiting the working places, so that they are fully acquainted with the details of the work going on. Each engineer computes the efficiency of all the jobs under his jurisdiction and the computations are then checked by the head of the department or the assistant.

The number of men required to carry on bonus work depends largely on the number of jobs. In turn, the number of jobs depends on the amount of attention which it is thought advisable to spend on the details of each major operation. For example, the shops may be given the job of completely overhauling a Mallet locomotive. A standard of 500 shifts might be set for the total job. This would be one job and would require very little work from the bonus engineer after the standard was set. The job might be divided into machine work, boilermaker work, steam fitting, and electrical work, making four jobs out of one, which would require practically four times as much work in the bonus department. Or, each individual operation on the overhaul job might be considered as a separate job, rates set, time kept, and the computations made on a great many details which would take considerable time on the part of the engineer. This last method of handling the bonus is the most accurate and is the method employed where the results obtained justify the cost of the clerical and engineering work. In some cases, it would cost as much to obtain an accurate standard on a particular job as it would cost to do the work. This is particularly true for work which is done once and is not repeated. In such a case no attempt is made to set a standard for the job. If the men doing such work have been regularly working on bonus, they are paid a flat rate of bonus for the time spent on the job. This does not furnish any incentive for faster work, but it prevents dissatisfaction on the part of the men required to do the work, and it is cheaper than spending the time to obtain a standard.

Some of the regular work about the plant is of such a nature that it is impractical to put it on a bonus basis, and such work is done on "company time," no bonus being paid over the daily wage rate. If a man works part of a shift on a bonus job and part of a shift on a company time job, he will be inclined to charge most of his time to the company time job, so as to increase his bonus. The boss must therefore keep the men's time carefully and accurately.

The bonus is figured on the first of each month for the work done the previous month, and the money earned is paid on the check the men receive on the 21st of the month. In case a man quits during the month, his bonus is mailed to him immediately after the bonus pay day.

We have found that in every case where a fair standard has been set for bonus work, it has resulted in lowering the cost of that operation, and that the men doing the work earned more money than before.

VENTILATION, MINE FIRES, AND SAFETY METHODS

Ventilation

The United Verde mine is mechanically ventilated throughout. This is necessary because of the poisonous sulphur fumes which are at times generated when blasting in the so-called "massive sulphides." It has also been found that control of air currents is sacrificed to some extent when natural and mechanical ventilation are combined. Positive control as to direction and volume is considered very important normally and absolutely necessary in case of a mine fire.

The virgin rock temperature on the 3000-foot level was 95°. Temperatures taken below this horizon in diamond-drill holes indicate a virgin rock temperature increasing about 1° for each 150-foot increase in depth.

The mine is ventilated by a main fan on the 1000 level. The air is drawn through an intake air shaft 13 by 13 feet in the clear and 980 feet deep. On the discharge side the air passes through a short duct which contains a multiple-blade damper for volume control, then down an inclined raise to the fresh-air shaft which extends downward to the 3000 level (see fig. 42). This shaft has an effective cross-sectional area of 80 square feet. Fresh air is taken off on all levels from the 1000 to the 3000 level. The volume is controlled on all levels by regulator doors which can be set for any desired volume.

The main return airway consists of a series of raises and connecting drifts of various lengths, beginning on the 2100 level and extending upward to the 900 level. At this point it connects with a shaft which extends to the 300 level. A drift to the surface completes the return air course. This course has an effective cross-sectional area of 120 square feet. Both the main fresh air course and the main return are free of timber or other combustibles throughout their entire length.

The main fan is of the double-inlet, reversible type, with forward curved blades. The rotor is 4 feet 6 inches wide by 9 feet 10 inches in diameter. It is direct-connected to a 400-hp., 2,200-volt, 280-r.p.m., 3-phase, 60-cycle, slip-ring motor. A 350-horsepower, 2,200-volt, 3-phase, 60-cycle, 350-r.p.m., wound-rotor induction motor is used as an auxiliary drive. This motor is belt-connected and can be called into service on a moment's notice by means of a clutch on the fan drive-shaft. Both motor controls are on one switchboard. A push button switch cuts the power from one motor to the other. The speed of both motors can be changed at will by means of a push-button type control, with a bank of resistances in the secondary circuit arranged to give three speeds. The multiple-blade damper was installed to give still greater volume control than is possible with the three motor speeds.

To obtain the maximum benefits from a ventilation system, it must be so arranged that there is a minimum of loss through short circuiting or leakage. Figure 41 shows a standard mine door on a motor level. In ventilating the United Verde mine, a constant check is kept on air currents to prevent loss of air into channels in which it becomes ineffective. Thus unavoidable losses in hoisting shafts are utilized on levels in the upper part of the mine for cooling purposes. Abandoned workings are sealed off in order to eliminate losses or air. Such economies in air are felt to be necessary if the man at the face is to derive the maximum benefits from the power expended at the fan.

No particular attention has been paid to the reduction of the mine resistance by smooth-lining main airways, but all new airways established in recent years have been made larger in order to take care of the resistance factor. A standard cross-sectional area of 120 square feet has been adopted.

In the course of mining it was deemed advisable to leave level pillars in certain sections of the mine. These pillars were later made use of as horizontal fire breaks. All through raises in these pillars were provided with iron doors hung in concrete frames (see fig. 43). When these raise cover doors are closed, an effective seal is established. This seal has been made use of in establishing a system of air splits. The ascending air column is not permitted to pass through the entire mine, but is withdrawn on the levels below the fire breaks and directed into the main return. Fresh air is distributed on the level above the break, thus assuring that each section of the mine is served by a split with reasonably

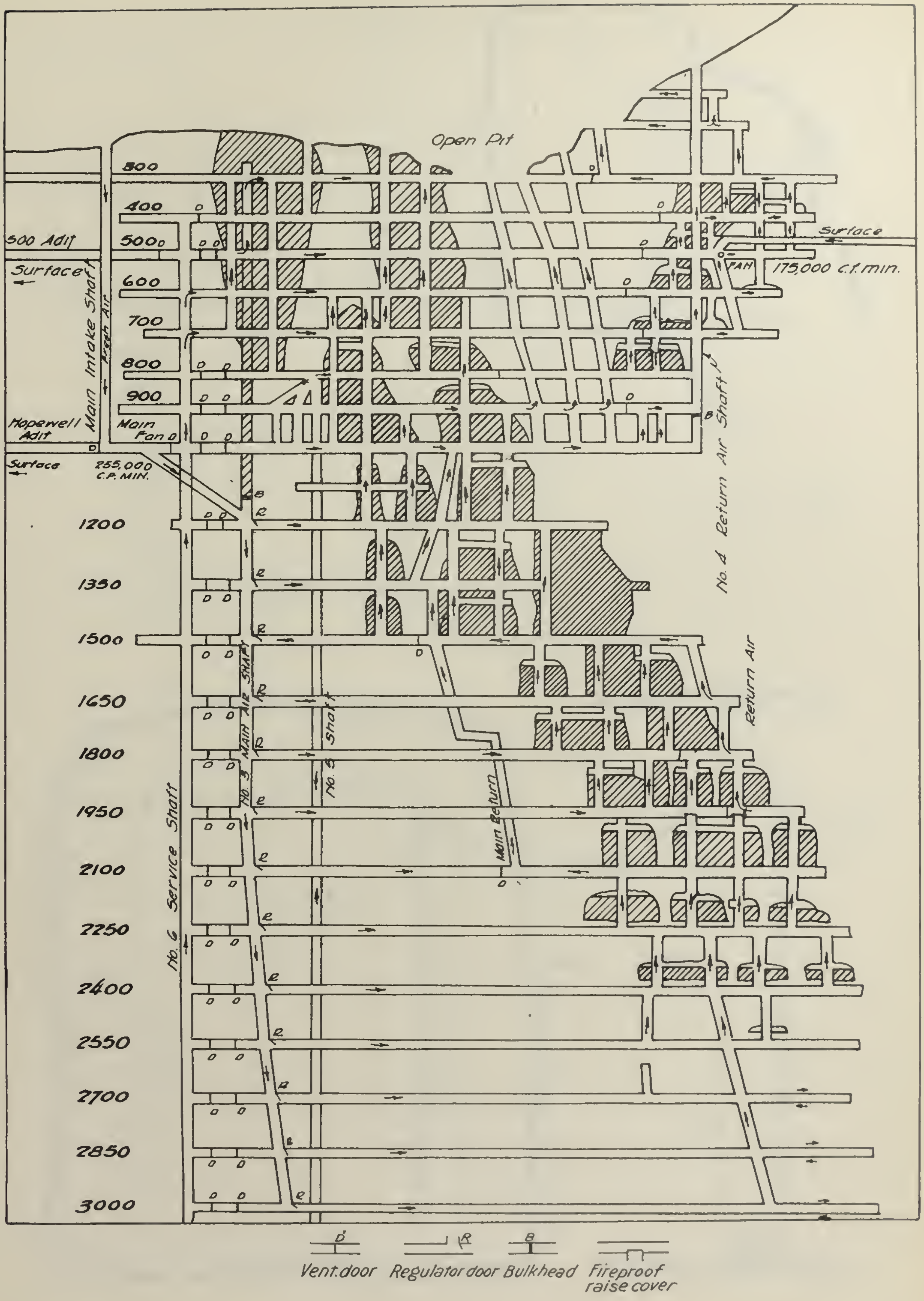


Figure 42.- Diagrammatic section, general ventilation system

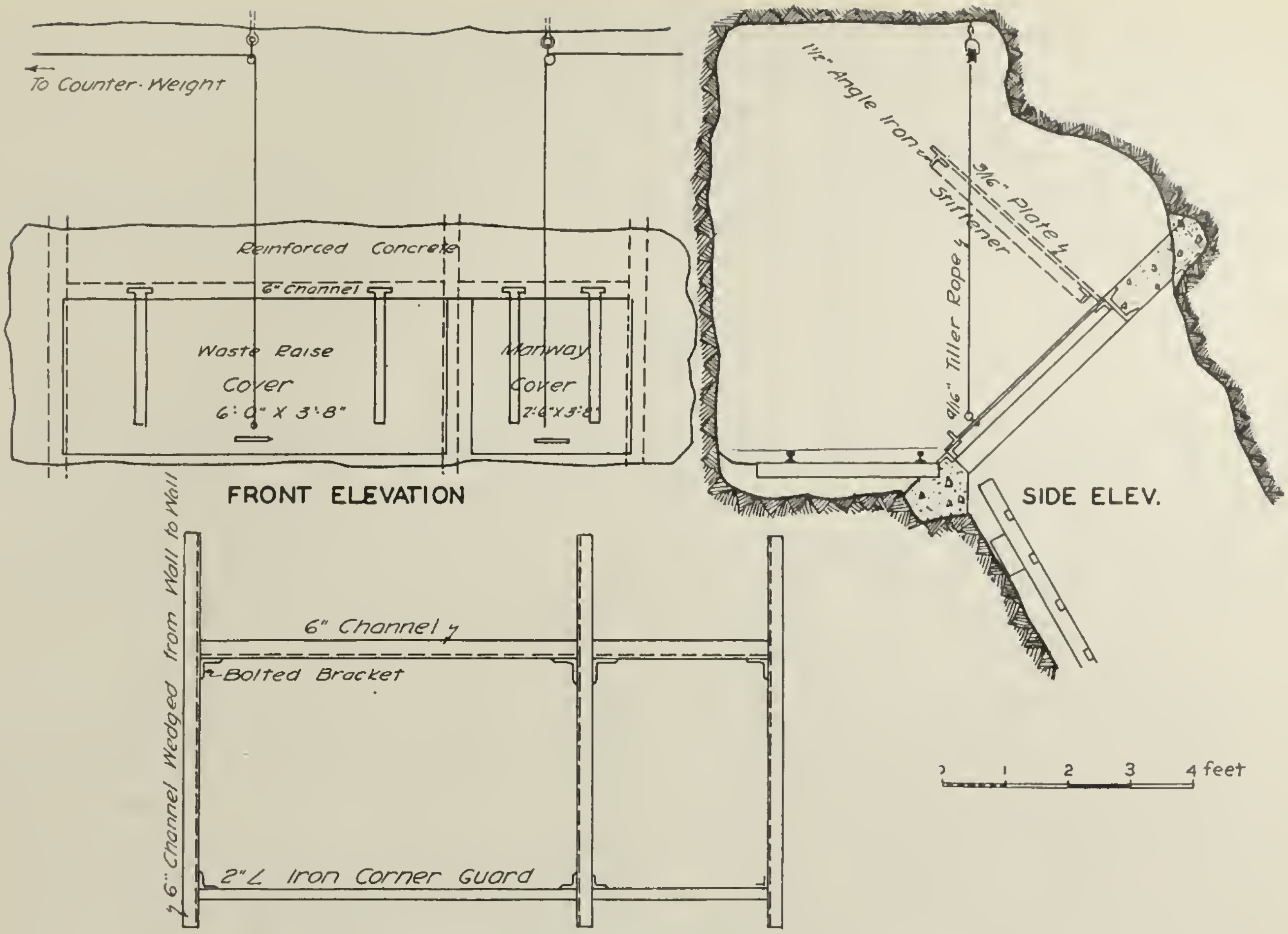


Figure 43.- Raise cover, showing doors and supports

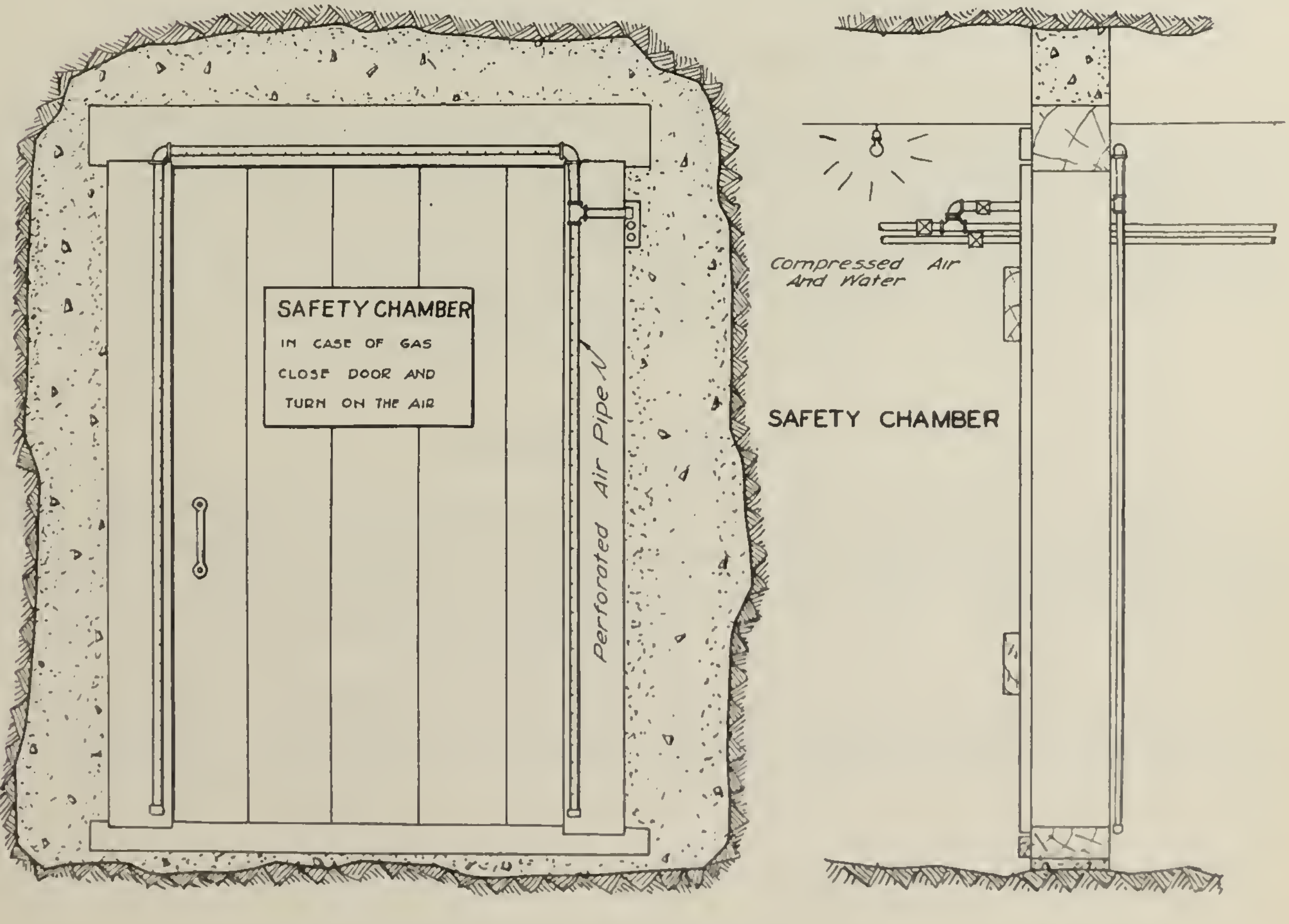


Figure 44.- Safety-chamber door

fresh air. There are, of course, isolated sections where small volumes of vitiated air pass from one split upwards into another. This is unpreventable in the course of mining, and has not been the cause for complaint because it has always been possible to withdraw such air without sending it through other active workings. Seals of the nature described exist on the 1950, 1500, and 1200 levels. In case of a fire, it is expected that the smoke can be controlled so that production will have to be stopped only on the levels immediately affected by the fire.

Safety Chambers

Some years ago safety chambers were of vital necessity for the safety of men underground. In more recent years their importance has not been so pronounced. The present method of blasting on signals, and the greater control now exercised over the air currents, gives a reasonable assurance of a safe exit. However, as an additional safety precaution, there are still maintained on each level one or more safety chambers into which men may retreat in any emergency and feel safe as long as the compressed-air lines are intact.

The safety chambers are located in dead ends of drifts. A standard-type mine door is installed as far from the face as space outside the chamber will permit (fig. 44). The chambers are supplied with electric lights, water, and compressed air lines. A sign on the door instructs men to close the door and turn on the air. Valves controlling both the air and water are inside the chamber. The escaping compressed air builds up sufficient pressure within to keep all gases out under normal conditions. As an additional precaution a 1-inch perforated air line is fastened on the outside of the door opening. Signs at various points on the level indicate the direction to the nearest safety chamber so that men can readily find them.

Auxiliary Ventilation Equipment

Auxiliary equipment is used to ventilate dead ends and development headings. This equipment consists of electrically-driven fans and compressed-air blowers of the injector type, connected to metal pipe or flexible tubing. The injector blowers are used only on very temporary jobs and where the air has not over 300 feet to travel. They are also used as line boosters, operating in series with an electrically-driven fan. Metal pipe is now only used in exhaust lines, and where heavy electric blasting is done.

Mine Fire Methods

The fire hazards in the United Verde mine are, with perhaps one exception, no different from those in any other mine. That exception is the heavy sulphide ore, the dust of which is known to cause explosions when permitted to accumulate. An explosion of this kind is very apt to cause trouble in a timbered stope.

Much is done to elininate fire hazards and to prevent fires. Fire patrolmen are employed who inspect not only the active workings, but spend considerable time in temporarily or permanently abandoned workings.

The gradual elimination of trolley wires is considered a move in the right direction toward the prevention of underground fires. Trolley motors are still in use on six levels, but 15 levels have storage-battery equipment.

Carbon tetrachloride fire extinguishers are located at all charging panels, and near all other types of electric equipment. Two and one-half gallon Foamite extinguishers have been placed on various levels close to heavily timbered areas. It is not expected that these extinguishers will be used to fight fires, but rather to put out incipient flames which might otherwise become serious fires. Ten fires of various kinds were extinguished between June and September, 1929, with hand-operated equipment. In addition to the above, the mine is equipped with a 5-inch water column to supply water for fires only. The compressed-air and water pipes on each level are connected to the column on the shaft stations, with valves so arranged that there is no interference in the flow of air or water under normal operating conditions. A blue print showing all connections and valve arrangements is kept under glass at each level station. Hose connections have been put in all water lines at 100-foot intervals where these pass through timbered areas. Special high-pressure hose in 50-foot lengths is kept in racks at strategic points on the levels. A 100,000 gallon capacity storage sump is located on the 1000 level. This is kept full of water from the main drainage system. A triplex pump, with a capacity of 159 gallons per minute is capable of pumping the water out of this sump to the surface storage or direct to any level desired. This system of utilizing the mine drainage water for emergency purposes has been found desirable because of the seasonal shortage of surface water.

Mine-Rescue Crews and Equipment

An active, trained rescue crew of 40 men is maintained. These men receive training once every two months in the use of oxygen breathing apparatus and gas masks.

Rescue equipment consists of 10 two-hour Gibbs breathing apparatus, 5 two-hour McCaa, and 3 one-half-hour McCaa breathing apparatus, and 20 Burrell's All Service Gas Masks. Two high-pressure oxygen pumps, one of which is mounted on a mine truck, one low-pressure fan, also mounted, and two fire-tool trucks, complete this equipment. One portable telephone is also used with this equipment.

Mine Safety

The safety work at the United Verde mine consists of inspections, education, safety advertising, and supplying approved safety equipment.

Every official is a safety inspector so that there is a constant inspection rather than only periodic inspection by one man. Inspections are nevertheless carried out periodically by the safety engineer. Safety committees chosen from among the miners and shop men make weekly inspections and report to their shift boss or foreman.

The principal mine accident hazards may be summarized as follows:

1. Fall of ground from roof and walls.
2. Flying of sulphide when collaring a drill hole or breaking rock with hammers.
3. Mine haulage.

4. Handling rock. This frequently causes severe cuts on hand and arms.
5. Sulphide gas generated when blasting in heavy sulphides.
6. Falls of men and material.

Safety education consists of lunch-hour talks with the men, bulletins, and photographs of the right and wrong way of doing work. Safety advertising consists of bulletins, pictures, blackboard announcements, and contests.

Safety equipment sold by the company at cost consists of "hard-boiled" hats, gloves, and respirators. Goggles are issued when men are hired. In addition to this equipment, all men are required to wear hard-toed shoes, which are sold by Jerome merchants. Safety belts and chains are issued upon request from the shift boss only.

The hard, brittle character of the ore makes it absolutely necessary that goggles be worn when starting a hole or when breaking rock with a hammer. Hard and fast rules to this effect have been made and must be observed on penalty of a lay-off. The hard-boiled hats and safety shoes which are required as a part of the miner's equipment have unquestionably prevented many serious accidents and, no doubt, a number of fatalities.

Foremen and shift bosses are held strictly responsible for the safety of the men under their supervision. They must see that the tools they use are in good repair, that the working place is safe, that the men observe all safety rules, and otherwise conduct themselves in the safest manner possible. The shift boss is expected to instruct the new man in his work, the hazards connected with it, and how best to eliminate or avoid them. The old employee is constantly reminded of the fact that safety should receive his first consideration on any work that may be assigned to him.

Certain rules are constantly stressed and impressed on old and new employees alike; violation of them means a lay-off or discharge from service.

A safety contest in the form of a horse race was started in July of 1929 and ended on December 31. Each shift boss and his crew was represented by a race horse. The horses were advanced several times a week and placed in their relatively correct position once each month. The position of any horse was dependent upon the frequency rate of lost-time accidents per 10,000 shifts. The crews without accidents were, of course, in the lead at all times. Seven horses tied for first place. A prize was given to each member of the winning crews. Another method used to stimulate interest in safety is that of giving out cigars at the end of each month to those crews which have gone the full month without a lost-time accident. The cigars are wrapped in foil and have printed on the wrapper:

In appreciation of your efforts
in Safety and Accident Prevention.
United Verde Copper Company.

The shift bosses too are rewarded for an excellent safety record. A monthly safety bonus is paid to them. An honor roll is posted each month, containing the names of those shift bosses who have gone three consecutive months or more without a lost-time accident to any of their men. Shift bosses having 10,000 man shifts to their credit without a lost-time accident receive a special award.

While there is some expense connected with this method of promoting safety, records show that it is very effective. It is believed that once the value of safety is firmly imbedded in the minds of the men, much in the line of contests and awards can be eliminated.

Figure 45 shows a comparison of lost-time accidents in 1928 and 1929.

MINE DRAINAGE

The Hopewell tunnel on the 1000 level, which is the ore-haulage level for the mine, is also the mine drainage tunnel. All mine water above this horizon is collected on the various levels and, by means of ditches and diamond-drill holes, passed from level to level until it reaches the Hopewell tunnel, through which it flows to the surface. The water collected below the 1000 level is lifted to the Hopewell tunnel by two pumps. Both are 6½ by 12 inch quintuplex pumps, with capacities of 500 gallons per minute each, lifting 1,000 feet vertically. They are geared to 150-horsepower, alternating-current motors. The main pump column is an 8-inch, lead-lined pipe, hung in the pipe compartment of No. 6 shaft.

The drainage water on the various levels is ditched to the main haulage crosscut and directed to within several hundred feet of No. 6 shaft, and there passed through a diamond-drill drainage hole downward to the next level. Thus the water between the 1000 and 1950 levels drains into the 1950-level pump sump; below the 1950 level it drains into th 3000-level sump. The diamond-drill drain holes are 2½ or 3 inches in diameter.

The pump on the 3000 level is operated about three hours per day and lifts its load to the 1950-level sump. The pump on the 1950 level runs about five and a half hours out of the 24, and discharges its water into the Hopewell tunnel drainage ditch.

Approximately 180,000 gallons of water flows through the Hopewell tunnel daily. On the surface this water is utilized in leaching operations on low-grade ores.

MINE ORGANIZATION

The administrative organization at the mine is shown in Figure 46.

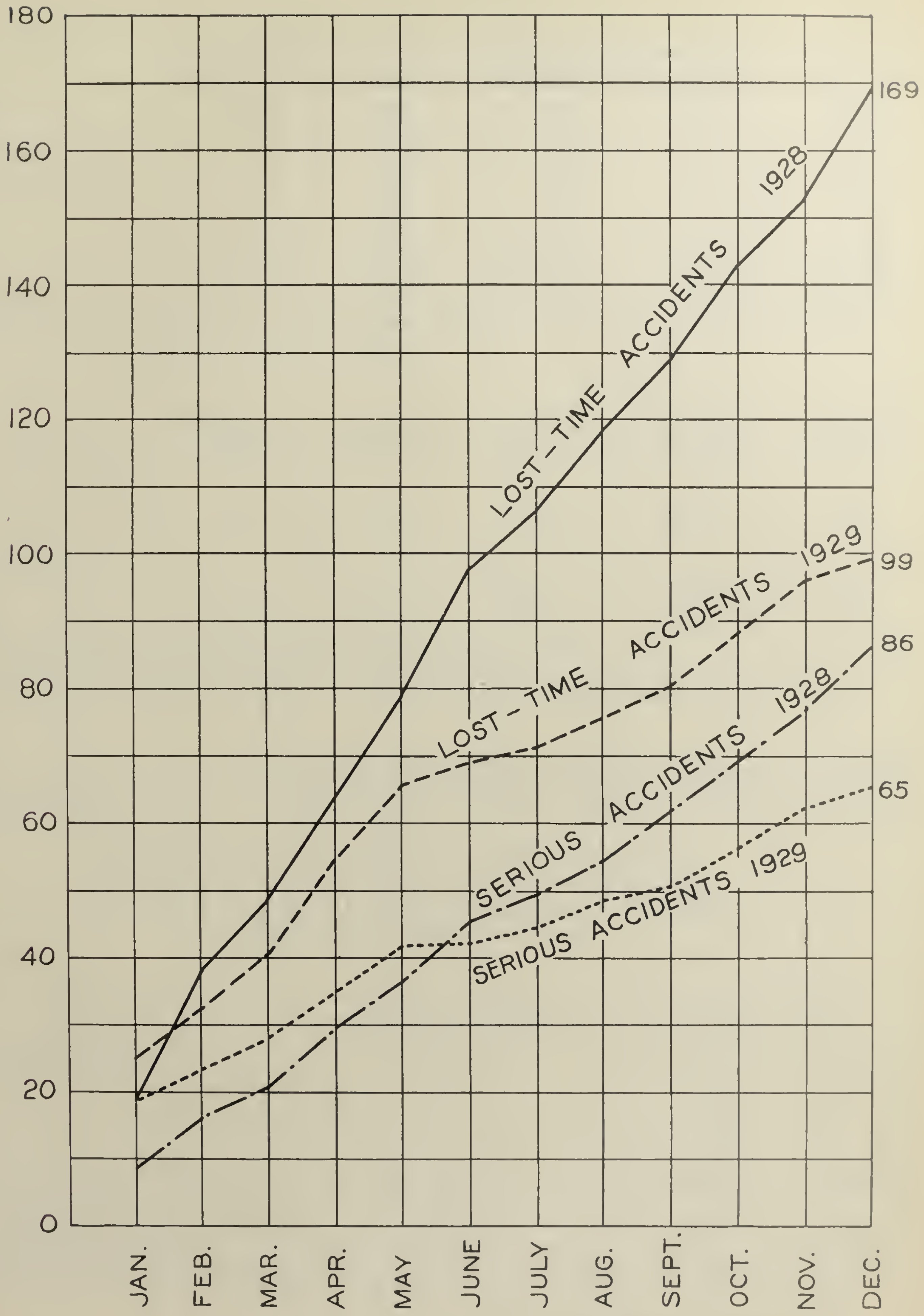


Figure 45.- Accident frequency curves, 1928 and 1929

	Development, hrs. per ton	Mining, hours per ton			
		Cut-and-fill	Square-set	Top slice	Shrinkage
Breaking	[1]0.157	0.135	0.383	0.383	0.248
Mucking	.119	.547	.872	.651	.559
Timbering and fill ..	– –	.466	.606	.837	.352
Haulage and hoisting.	– –	.238	.238	.234	.260
Supervision	.006	.122	.122	.120	.134
General	– –	1.037	1.039	1.020	1.136
General	– –	1.037	1.039	1.020	1.136
Total	0.282	2.544	3.260	3.245	2.689

1 Development figures show hours on development against total stope tonnage; 0.157 hours per ton is for both breaking and timbering.

	Development[1]	Cut-and-fill	Square-set	Top slice	Shrinkage
Average tons per shift	28.33	3.14	2.45	2.47	2.97
Labor, per cent total time	9.19	53.01	28.08	8.32	1.40
Powder, pounds per ton[2]	.45	.55	.60	.71	.71
Timber, board feet per ton	.83	6.95	12.52	9.91	2.36
per ton	.83	6.95	12.52	9.91	2.36

1 Development figures are computed against total stope tonnage.

2 Principal powder used is 1⅛ by 8 inch, 50 per cent gelatin.

Average tons per man-shift on surface charged to underground operations is 74.95.

	Air compression	Hoisting	Pumping	Ventilation	Lighting
Power kw.h. per ton	10.15	2.35	0.26	3.07	0.61

Development details in units of labor, power, and timber.
Average drift and crosscut is 6 feet wide by 8 feet high.
Average raise is 6 by 11 feet with one timbered compartment.
The rock is generally hard and firm.

Labor	Drifts and crosscuts, hours per foot	Raises, hours per foot	Total
Breaking and timbering	3.50	9.15	4.75
Mucking	4.17	1.56	3.59
Supervision	.19	.19	.19
Supervision	.19	.19	.19
Total	7.86	10.90	8.53
Feet advance per 8-hour shift	1.02	.73	.94
Powder, pounds per foot[1]	13.56	14.36	13.74
Timber, board feet per foot	4.57	97.02	25.03

1 Principal powder used is 1⅛ by 8 inch, 50 per cent gelatin.

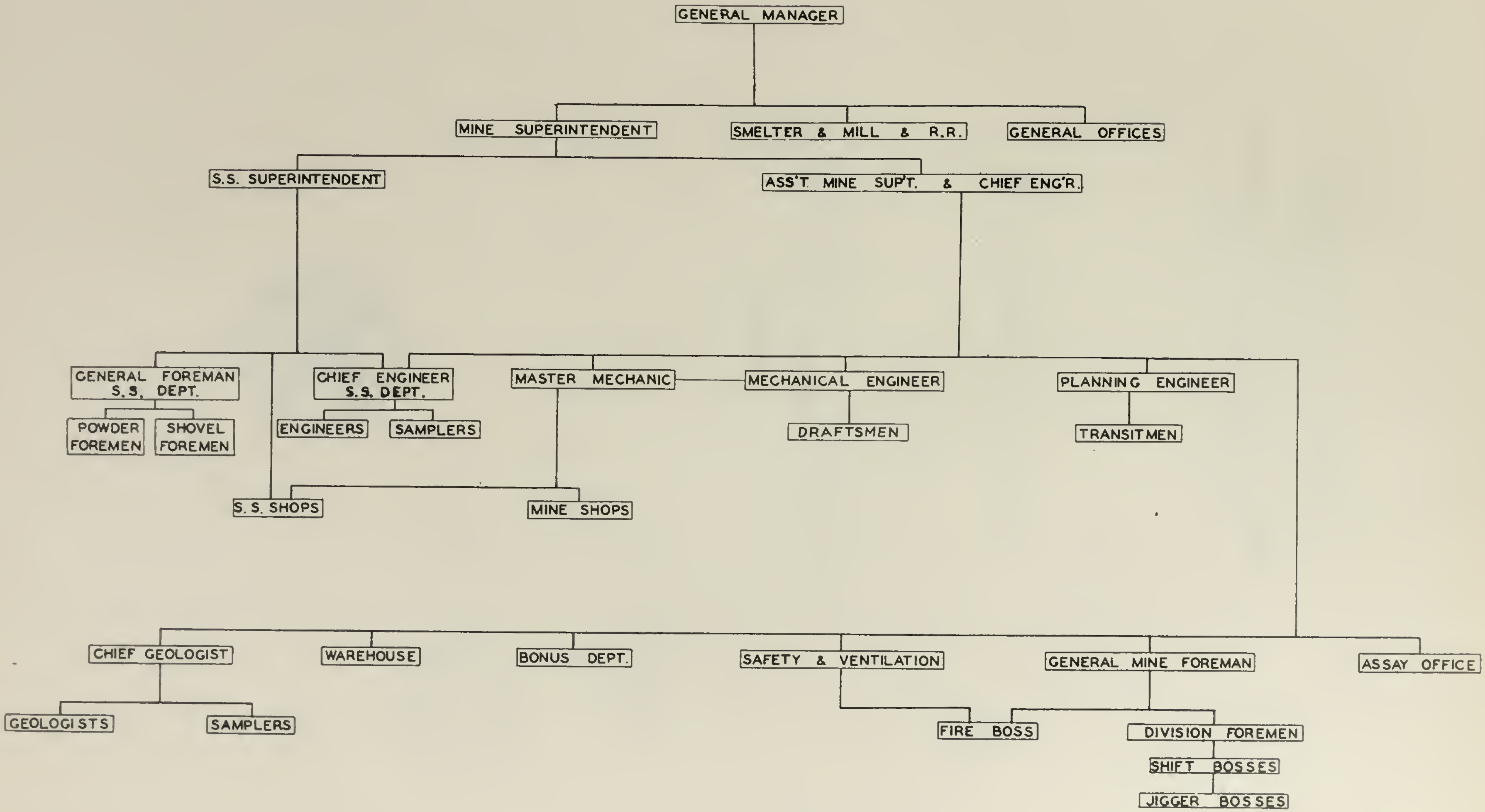


Figure 46.-Organization chart

I. C. 6441

January, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

GEOPHYSICAL ABSTRACTS

NO. XXI

BY

F. W. Lee

I.C. 6441.
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DEPARTMENT OF COMMERCE -- BUREAU OF MINES

GEOPHYSICAL ABSTRACTS[1]

No. 21

Compiled by Frederick W. Lee[2]

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6441."

2 - Senior physicist, U. S. Bureau of Mines.

1. GRAVITATIONAL METHODS

(1) UN INSTRUMENT TRANSPORTABLE POUR LA MESURE RAPIDE DE LA GRAVITÉ

(A PORTABLE INSTRUMENT FOR QUICK MEASUREMENT OF GRAVITY)

By F. Holweck and P. Lejay

Comptes Rendus de l'Académie des Sciences, Paris, vol. 190, No. 24, 1930, pp. 1387-1388.

The principles and the construction of the instrument are described. The ensemble of the instrument consisting of two supports provided with levelling screws and levels, two cells, two amplifiers, registering device, dry pile batteries, and four interchangeable pendulums is packed into two small boxes, the whole weight of which does not exceed 40 kilograms. Only a few minutes are required to put the instrument in operation.

Preliminary measurements carried out by the Observatory of Paris during several months have shown that the variations of the elasticity of invar during this time was negligible. The clamping device by which the pendulum is rendered immovable during transportation preserved it well from any injury.

A table shows the periods obtained for the two pendulums from the tests carried out along the base-line Paris-Dunkirk.---W. Ayvazoglou.

(2) DAS PENDEL MIT OSZILLIERENDEM AUFHÄNGEPUNKT

(A PENDULUM WITH OSCILLATING POINT OF SUSPENSION)

By Paul Hirsch

Zeitschrift fuer angewandte Mathematik und Mechanik, vol. 10, No. 1, 1930, pp. 41-52.

The problem, which is not new, is treated by means of classical mechanics; in each phase, clearness is sought for. If the point of suspension of a pendulum is given a vertical harmonic movement with small amplitude and great frequency, the stability of the lower position of equilibrium increases and the lability of the upper decreases and in case of sufficiently strong oscillation changes into stability. The conditions change if the general case is not considered, but the oscillation is assumed to be in a direction inclined to the vertical. In this case there are two stabile and two labile positions of equilibrium determined by the roots obtained from an equation of the fourth degree; only one value for each position exists in case the oscillation is weak.

In case of vertical oscillation the two labile and one stabile positions of equilibrium coincide in the upward direction. Preliminary calculations on the effect of a mathematical pendulum are made; two generalizations are

discussed: 1. Elliptical oscillation. The point of suspension of the pendulum is supposed to perform harmonic oscillation of equal frequency in two directions--that is, it describes an ellipse. In this case the pendulum has usually six positions of equilibrium. 2. Linear oscillation of a physical pendulum is a symmetrical position. A generalization results from the discussion. The following cases are examined: 1. Linear oscillation of a physical pendulum. 2. Circular oscillation. For this special case of the elliptical oscillation there results a duality with the linear oscillation--that is, a conformity of the positions of equilibrium if the direction of the linear oscillation is interchanged with that of the line perpendicular to the plane of the circular oscillation and the words "stabile" and "labile," as well as "upper" and "lower", are substituted one for another. 3. Elliptical oscillation. Linear oscillation which does not proceed harmonically but with a constand velocity and a sudden reversal of it can be easily treated in the same way. Only the results obtained in this case are given. The direction of all the positions of equilibrium can be calculated by means of a simple approximate proceeding. The frequency with which the pendulum can oscillate around the stabile positions of equilibrium is explained.--Author's abstract translated by W. Ayvazoglou.

(3) DIE INVARIABILITÄT UND ABSTIMMUNG VON MINIMUMPENDELN

(THE INVARIABILITY AND ADJUSTMENT OF MINIMUM-PENDULUMS)

By E. Kohlschütter

Zeitschrift fuer Geophysik, vol. 6, No. 8, 1930, pp. 466-476.

The question raised by O. Meisser concerning the development of a formula for the variability of invariable minimum gravitational pendulums produced by small knife-edge displacements or reasons having a similar effect is formulated and applied to pendulums which can be manufactured practically. A table for this variability is calculated. Formulas for calculation and adjustment of cylindrical minimum-pendulums are derived.--Author's abstract translated by W. Ayvazoglou. Note: The article is followed by two brief discussions, one written by O. Meisser under the heading: Bemerkung zu der Arbeit "Die Invariabilität und Abstimmung von Minimum-pendeln" by E. Kohlschütter (Remarks on E. Kohlschütter's work "The Invariability and Adjustment of Minimum Pendulums); the other is an "Answer" (Erwiederung) written by E. Kohlschütter.

(4) NUOVI CALCOLI DI INFLUENZE GRAVIMETRICHE DI TERRENI SUPERFICIALI E PROFONDI

(NEW CALCULATIONS ON THE GRAVIMETRICAL INFLUENCE OF SUPERFICIAL AND DEEP MASSES)

By Arnaldo Belluigi

Ergänzungshefte fuer Angewandte Geophysik, vol. 1, No. 2, 1930, pp. 141-148.

The author establishes new calculations on the gravimetric influence

of superficial and subsurface disturbing masses. He refers to some of his previous investigations concerning topographic correction and develops a new process in connection with it. The author tries to solve the question of the influence of disturbing masses at a depth by approximation of imaginary disturbing bodies to the real ones. A special solution for an imaginary body of the form of a parallelepiped is given.--Author's abstract translated by W. Ayvazoglou.

(5) VERGLEICH DER AUFZEICHNUNGEN ZWEIER GALITZINPENDEL MIT VERSCHIEDENER EIGENPERIODE

(COMPARISON OF RECORDS OF TWO GALITZIN PENDULUMS WITH DIFFERENT NATURAL PERIODS)

By Helmut Landsberg

Gerlands Beiträge zur Geophysik, vol. 27, No. 3-4, 1930, pp. 326-359.

The same component was registered in the Taunus Observatory by two Galitzinpendulums, one with the natural period of 3 seconds and the other of 18 seconds. The records of these pendulums, already discussed by B. Gutenberg in his article "Registration made with two Galitzin-pendulums with different periods," (see Geophys. Abs. No. 13, p. 11), were submitted to a systematic investigation. The advantage of the installation of the instruments consisted of the fact that the starting of the earthquake was very distinctly recorded by the pendulum with short periods; clear records of the after-shocks were obtained by this pendulum also. Ninety-four earthquakes were investigated and it was proved that waves of long periods never arrived before the waves of short periods.

In more than 50 per cent of all the cases there were no waves of long period in P; they appeared there only in the later phases. The possibilities of the origin of a time difference of from one-half to 2 seconds in P of both kinds of waves are discussed. The phase maximum of long-period pendulum in P was generally delayed if compared with the short-period pendulum. The ratio of maximum amplitudes of the P-phase in both pendulums shows a distinct dependence on the focus-distance.

There were unknown groups of short waves observed between P and S during most earthquakes; they were used for the construction of some travel-time curves, of course with restrictions. A general comparison of the appearance of long and short waves at single phases of earthquakes was carried out and it was proved that almost all the groups which were longitudinal on their whole course have shown waves of short periods. The question of the appearance of short periods in changing waves is discussed, and a general view on the proportion of the different phases in the two pendulums is given.

Microseisms of only 4 to 10 seconds were investigated because only these earthquakes were recorded by both pendulums. A dependence of the periods in the two pendulums could not be established. Concerning the yearly course of the periods, it was found that the long and the short periods were recorded at the same time in summer as well as in winter, and only the amplitudes which

belong to the single waves were changed. The influence of the English and Norwegian surf was investigated, and it was again established that microseisms and surfs always change in the same sense.--Author's abstract translated by W. Ayvazoglou.

(6) GEOLOGICAL INTERPRETATION OF MAGNETOMETRIC AND GRAVIMETRIC OBSERVATIONS WITH THE AID OF APPARATUS FOR MECHANICAL CALCULATIONS (IN RUSSIAN)

By G. Gamburtzeff

Journal of Applied Physics, Moscow,
vol. 5, No. 3-4, pp. 227-235, and vol. 6, No. 1, pp. 62-67, 1929.

The author describes mechanical instruments by which the deviations of the magnetic potential and gravimetric potential of an infinitely long homogeneous cylinder of any section can be calculated. The whole course of the distribution curve of the force of gravity perpendicular to the peripheral line of the cylinder can be graphically represented directly by encircling the borders of the cross section of the cylinder.

Owing to the well-known relation between the magnetic potential and gravitational potential (in case of a homogeneous magnetization) the apparatus can serve for geological interpretation of both gravimetric and magnetic observations. Theoretical calculation for constructing the instruments and graphical designs of them are given.--W. Ayvazoglou.

2. MAGNETIC METHODS

(7) BEZIEHUNGEN ZWISCHEN GEOLOGISCHEN UND ERDMAGNETISCHEN VERHÄLTNISSEN

(RELATION BETWEEN GEOLOGIC AND EARTHMAGNETIC CONDITIONS)

By F. W. Pfaff

Zeitschrift fuer Praktische Geologie,
vol. 38, No. 9, pp. 129-135 and No. 10, pp. 154-159, 1930.

Wettergebirge has been chosen by the author for investigation of the way in which the earthmagnetic values of declination, horizontal intensity, and inclination are affected by the geological structure of the upper earth layers. The best way to understand the correlation between the geological and magnetic conditions is to compare the values of the magnetic elements which can be expected under normal conditions with those obtained by measurements. In a table accompanying this article Pfaff gives the measured and normal values of the magnetic elements as well as the differences between these values for a few formations. At the end of the table the mean values of the differences are calculated for shell limestone, belemnite limestone, raibler and haupt dolomite.

The second part of the article deals with influences upon the values of the magnetic elements in the region described in connection with:

The faults and lines of disturbances,

The influence produced by a syncline,

The reduction for a level terrane,

The explanation of the small values for H and I in the northern Mittenwald,

The influence caused by the great lines of disturbance.

In conclusion the author says that, if the measured values only are compared, the direct influence of the geological conditions upon the values of the magnetic elements can hardly be noticed, but they become noticeable when corrected (specific magnetization) for the topographic and geographic positions, as well as for the thickness of the layers (geological and tectonic height); thus the following conclusion has been drawn: For most of the stations of the region in which the measurements were carried out, the specific magnetization agreed with the tectonic height.--W. Ayvazoglou.

3. SEISMIC METHODS

(8) PROGRESS OF SEISMOLOGICAL INVESTIGATIONS IN THE UNITED STATES

By N. H. Heck

U. S. Coast and Geodetic Survey, Special Publication No. 167, 14 pp.

In this pamphlet the author gives a brief description of the progress of seismological investigations in the United States covering a period from July 1, 1927, to January 1, 1930. Attainments made during this period affected the following problems of seismology:

1. Installation of high-grade instruments;
2. Development and perfection of new types of instruments;
3. Expansion of programs of local investigations in regions more frequently subject to earthquakes;
4. Expansion of programs of teleseismic investigation;
5. Attack on the engineering, insurance, and other practical phases of the problem;
6. Unprecedented public interest in the subject.

The pamphlet is divided into two parts; the first deals with the work of the Coast and Geodetic Survey; in the second a summary of earthquake investigations in the United States made also by other institutions is given.

Two maps showing (1) the seismological stations of the United States

and adjacent Canada and (2) important seismological stations in the United States and regions under its jurisdiction are added.

A list of teleseismic stations distributed in the United States concludes the article.--W. Ayvazoglou.

(9) SEISMISCHE UNTERSUCHUNGEN AUF DEM PASTERZEGLETSCHER

(SEISMIC INVESTIGATIONS ON THE PASTERZE GLACIER)

By B. Brockamp and H. Mothes

Zeitschrift fuer Geophysik, vol. 6, No. 8, 1930, pp. 482-500.

The authors describe the seismic investigations carried out by them in August, 1929, on the Pasterze Glacier (Eastern Alps). The purpose of the work was as follows:

1. To make seismic measurements at fixed places of the glacier basin.

2. To collect material, following the direction of the work established by the Geophysical Institute of Göttingen, for the study of special questions connected with the propagation of waves.

Determinations on the following waves were made:

1. Longitudinal waves, V_p = 3,580 meters per second.

2. Transversal waves, V_s = 1,670 meters per second.

3. Longitudinal waves which passed through the rocks situated below the glacier, V_{pl} = 5,850 meters per second.

4. Longitudinal and transversal waves which passed through the ice at the lower border of the ice, U_p and U_s.

5. Reflected longitudinal waves, R_I and R_{II}.

The P^1-waves and the U_p-waves were used for the determination of the mean depth, and the R_I-waves were used for calculation of single values of the depth.

They served for the construction of longitudinal profiles and cross profiles of the glacier.

A series of figures showing these profiles is given. More than 70 seismograms were taken. Nine of them are shown in the figures.--W. Ayvazoglou.

(10) DAS IMPULSFELD DER PRAKTISCHEN SEISMIK IN GRAPHISCHER BEHANDLUNG

(THE IMPULSE FIELD OF PRACTICAL SEISMICS TREATED GRAPHICALLY)

By E. A. Ansel

Ergänzungshefte fuer angewandte Geophysik,
vol. 1, No. 2, 1930, pp. 117-136.

The author derives graphical methods by which the strata of the subsoil, the inclination of the strata and the discontinuities of the strata can be determined in more or less simple cases with the aid of the travel-time curve.

The great importance of the "contact curve" for the graphical treating of the impulse field is mentioned, and the construction of this curve from the data obtained for the travel-time curve is explained.--W. Ayvazoglou.

(11) ERGEBNISSE SEISMISCHER UNTERSUCHUNGEN UEBER DEN
SCHICHTENAUFBAU VON NORD DEUTSCHLAND

(RESULTS OF SEISMIC INVESTIGATIONS MADE OVER THE STRUCTURAL
LAYERS IN NORTH GERMANY)

By O. Barsch and H. Reich

Ergänzungshefte fuer Angewandte Geophysik,
vol. 1, No. 2, 1930, pp. 165-188.

The article is divided into the following three parts: (1) General remarks. (2) Seismic investigations near Dobrilugk. (3) Seismic investigations in Priegnitz and Schleswig-Holstein.

Diagrams showing the travel-time curves and the seismograms obtained from the investigations in the regions mentioned above are given.

Based on the results from this work in various regions of North Germany the authors determined the values for the velocities of wave propagation in a series of rocks:

Dry sandy diluvium	700 to 1,000 meters per second.
Watered diluvium Marly diluvium Sandy-clayer Miocene Clayey-sandy Oligocene	1,000 to 1,700 meters per second.
Marley Eozoic (?)	1,800 meters per second.

Chalk - - - - - - - - - - - - - - - - 2,100 to 2,250 meters per second.

Productive carbon (clay slate and sandstone - - - - - - - - - - - - about 3,800 meters per second.

Cambrian (clay slate and quartzites) about 5,000 meters per second.

A few other layers with V = 3,000 to 3,600 meters per second, found at different depths, are mentioned but with much caution. In spite of their reserve, the authors seem to be of the opinion that the deeply situated layers of discontinuity, which for all the profiles given in the article have an almost horizontal position, may be considered transgressive layers. While such a transgressive layer, covered with chalk, was found in Priegnitz at about 800 meters in depth, in Schleswig-Holstein the chalk was missing and the depth of the transgressive layer was only about 600 meters.

The real reason for the great magnetic and gravimetric disturbances in North Germany could not be established with certainty by seismic methods. Therefore it may be concluded that these rocks, at least as far as the regions investigated are concerned, are probably situated at a depth greater than 1,000 meters.

The well-known subterranean structure of the region under investigation was of great assistance to the authors in studying the course of the seismic waves, as well as other problems of applied seismics important from both practical and theoretical viewpoints.--Author's abstract translated by W. Ayvazoglou.

(12) À PROPOS D'UNE ONDE LONGUE DANS LA PREMIÈRE PHASE DE QUELQUES SÉISMOGRAMMES

(CONCERNING ONE LONG WAVE APPEARING IN THE FIRST PHASE OF SOME SEISMOGRAMS)

By O. Somville

Gerlands Beitrage zur Geophysik, vol. 27, No. 3-4, 1930, pp. 437-442.

In this article the author mentions a very striking long wave of a period of about 24 seconds which was registered in some seismograms obtained in the Observatories of Taunus, De Bilt, and Uccle by means of Galitzin pendulums with long natural periods during the earthquakes observed in Italy in 1928, 1929, and 1930.

This wave, which the author calls the PL wave, has so far been observed only in connection with the Italian earthquakes. Thus, before trying to give an interpretation on the nature of this wave the author raises the following questions: Can the PL-wave be observed in other directions than the northern, as in Italian earthquakes? Has it been observed in other cases? He adds that, of course, the answer to the questions may be difficult, as there are only a few stations working with seismographs with long natural periods.

In the last figure the author has drawn, based on observations

mentioned in this article, a first curve of times of propagation of the PL-wave for distances from $\triangle$ = 500 kilometers to $\triangle$ = 1,400 kilometers.--W. Ayvazoglou.

4. ELECTRICAL METHODS

(13) RESISTIVITY MEASUREMENTS OF OIL-BEARING BEDS

By F. W. Lee and J. H. Swartz

U. S Bureau of Mines, Technical Paper 488, 1930, 12 pp.

In the introduction to this paper the authors propose to show the results of experiments in oil-bearing territory, using electrical resistivity methods. The development of these methods is briefly mentioned.

Under the heading of "Partition Theory" two methods of approach for determining the character of subterranean formations are evolved. In the first class the authors place a theoretical discussion of parallel beds with different resistivities in each one, as discussed by Hummel, and in the second the results of field observations without complete mathematical analysis.

To test this theory a district in Allen County, Ky., was selected. The oil here is at an extremely shallow depth and therefore all conditions are favorable for discovering oil as outlined in the article. A geological description of territory is given and factors controlling the accumulation of oil in this area, as well as the groups into which the oil sands of Allen County fall, are mentioned. The process of taking measurements and apparatus used for observations (Rooney-Gish apparatus) is described. The contour of the testing site is shown in a figure.

The results of measurements carried out at three stations are illustrated by diagrams. The work was carried out in December, 1929.

In conclusion the authors say that although the results of these tests show favorable reactions under certain circumstances, they require much more work to establish their validity; thus this paper may serve as an experimental guide to persons interested in locating oil at shallow depths, and continued application of these methods is necessary to improve their technique before very deep investigations of oil formations can be made with reliable results.

A list of books relating to electrical resistivity methods is added.--W. Ayvazoglou.

(14) THE USE OF GEOELECTRICAL METHODS UNDERGROUND

By G. P. Haller

The Mining Journal (Phoenix, Arizona), vol. 14, No. 9, 1930, pp. 11 and 30.

In this article the author discusses the application of geoelectrical methods of prospecting to underground investigation by which the miner is given

a chance to obtain more definite information as to the presence of ore before further mining exploration is begun.

The underlying principles of the method are mentioned briefly and the effects obtained by geoelectric measurements in different cases by which the determination of the location of existing ore bodies is possible are shown in a series of figures. The necessity of testing the conductivity of samples of the minerals which occur in the area to be investigated before starting the actual survey is mentioned. Illustrations obtained from geoelectrical investigations of copper mines accompany the article.

In conclusion Haller says that the real benefit to be derived by the mining engineer and the geologist from this branch of geoelectrical investigation is evidenced by practical experience, but he adds that although of great value in capable hands the results can be more damaging than helpful in the hands of the inexperienced, as only a thoroughly trained expert in geoelectrical methods is able to compute the results and to interpret the obtained data geologically and with exactness. Furthermore, a close cooperation with the geologist is essential.--W. Ayvazoglou.

(15) GEOPHYSICAL STUDY PREDICTS ROCK CONDITIONS AT TUNNEL SITE

By E. E. Carpenter and E. G. Leonardon

Reprint from Engineering News-Record, September 4, 1930.

Lack of knowledge of underground conditions at the sites of certain engineering works, especially dams and tunnels, affects construction methods and costs to such an extent that advance information in regard to rock formation and water conditions is of the utmost value.

No attempt is made in this article to detail the technique of the geophysical exploration, and only a brief explanation of the theory of rock resistivity upon which the electrical method is based is given.

The authors describe the electrical exploration work at Bridge River Tunnel carried out in the summer of 1928.

The constitution of the rocks encountered along the tunnel which have been bored through since are indicated, proving the accuracy of the results obtained by examination of the resistivities of the rocks.--W. Ayvazoglou.

(16) ON THE RESULTS OF THE ELECTRICAL PROSPECTING IN THE REGION OF NOVO-GROZNY (IN RUSSIAN)

By A. Shaiderov

Azerbeidjanskoe Neftianoe Khoziaistvo, vol. 10, No. 9, 1930, pp. 78-85

After a brief explanation of the principles of electrical methods of prospecting, Shaiderov describes the methods carried out for the electrical investigation of the geological structure in the region of Grozny in the year 1929-30.

The following methods were applied:

1. The resistivities of rocks deposited at certain depths were determined by construction of electrical profiles.

2. The resistivity of rocks at different depths was determined by electrical boring.

3. The extension of rocks and their inclination was determined by applying stakes.

For the construction of profiles corresponding to different depths three different distances (150, 600, and 1,000 meters) between the electrodes were applied.

Based on the favorable results of investigation, which agreed entirely with the known geological data of the region, the author draws the conclusion that electrical methods of prospecting can be applied with advantage in case the horizons under investigation have distinct electrical characteristics.

The article is illustrated by the following maps:

1. A map showing the character of the electrical profiles for distances of 150 and 1,000 meters between the electrodes.

2. A map showing the results of electrical measurements at a depth of from 150 to 200 meters.

3. A map showing the results of electrical measurements at a depth of from 20 to 30 meters.

4. A section and profile along the line of shafts.

5. A map of vectors of inclination of the rocks.

W. Ayvazoglou.

(17) RESULTS OF SOME RECENT GEOPHYSICAL TESTS

By L. H. Henderson and V. P. Pentegoff

Reprint from The Mining Journal, Phoenix, Arizona, 3 pp.

Based on the accompanying maps and figures, the authors discuss a series of problems solved by electrical methods of prospecting:

1. Map showing resistivity survey of Silver Monument Mine, Chloride, N.M.

2. Map of detailed electrical survey for Spruce Mountain Mining Co., Sprucemont, Nev.

3. Details of the electrical survey of the Silver Plume property of Minaret Consolidated Mines Co., Cananea, Sonora, Mexico.

4. Map showing area in which electrical survey was conducted at Chino Mines, Nevada Consolidated Copper Co., Santa Rita, N. M.

5. Comparative profiles of results from electrical survey and drill prospecting at the Nevada Consolidated Copper Co., Chino Branch.

In all the cases the results of geophysical surveys were very satisfactory and agreed with the data obtained by drilling.--W. Ayvazoglou.

(18) AMPLIFICATORI GEOMETRICI DI PICCOLE DEFORMAZIONI DI LINIE DI CORRENTE IN UN SUOLO ARTIFICIALMENTE ELETTRIZZATO

(GEOMETRIC AMPLIFIER OF SMALL DEFORMATIONS OF LINES OF CURRENT IN AN ARTIFICIALLY ELECTRIFIED SOIL)

By Arnaldo Belluigi

Ergänzungshefte fuer Angewandte Geophysik, vol. 1, No. 2, 1930, pp. 137-140

The author gives the theory of a geometrical "amplifier" by which the small deformations of the lines of current in an artificially electrified soil can be disclosed.

He concludes that these lines can be used for the interpretation of the geoelectrical results.--Author's abstract translated by W. Ayvazoglou.

(19) EINIGE ÜBERSCHLAGSRECHNUNGEN ZU DEN PHASENVERHÄLT-NISSEN IM POTENTIALFELD BEI GEOPHYSIKALISCHEN BODENUNTERSUCHUNGEN MIT WECHSELSTROM MITTLERER FREQUENZ

(SOME CALCULATIONS CONCERNING THE PHASE-RATIO IN A POTENTIAL FIELD DURING GEOPHYSICAL INVESTIGATION OF THE SOIL WITH ALTERNATING CURRENT OF MEDIUM FREQUENCY)

By W. Heine

Ergänzungshefte fuer angewandte Geophysik, vol. 1, No. 2, 1930, pp. 156-164

With the aid of the approximate formula derived previously, the author calculates the phase displacement between two lines of current in media of unequal electrical conductivity for different values of conductivity.

The dependence of the phase displacement on the absolute values of the specific gravities of the two media is proved.

The difference in phase displacement between the lines of current placed at 1 and 10 meters distant from the deposit, the effect of this difference for measurements of the potential lines being the highest in case of minimum width, obtains higher values only if one of the two media has a specific resistance of the order of from 10^5 to 10^6 ohms, while in case of lower or higher values this phase difference remains very small. This agrees

with experience. By using an approximate proceeding the author derives and calculates also the phase displacement resulting from the change of current density along the central perpendicular line drawn on the line connecting the electrodes ("zero" potential line).--Author's abstract translated by W. Ayvazoglou.

(20) DAS ELEKTRISCHE UND MAGNETISCHE FELD UM EINEN "ERDSTRAHLER"

(THE ELECTRICAL AND MAGNETIC FIELD AROUND A "DIPOL")

By Herbert von Ludwiger

Ergänzungshefte fuer angewandte Geophysik, vol. 1, No. 2, 1930, pp. 189-226.

Contents of the article:

Introduction: Purpose of the work.

Description:

Chapter 1. The elliptically polarized alternating field examined at one point of the field measured.

Chapter 2. The measurements, the terrane on which the measurements were made, and apparatus.

Chapter 3. Summarized representation of the measured data.

Chapter 4. Physical meaning of the curves.

The article deals with measurements of the electromagnetic alternating field around a dipol in a medium of poor conductivity. In case of alternating currents of the tone-frequency character, the normal field at the point of measurement was found to be an elliptically polarized rotation field.

Thus the measurement proceeding required the taking into consideration of the time necessary for the flow of the current within one period of the alternating current produced. Therefore, at each point of measurement, the electrical field inside of the soil, as well as the magnetic field above the soil, was measured for each series of azimuths, with regard to the amplitude and phase, for two quadrants around a dipol of one-fourth square kilometer; the voltage branched off from the excitation served as a comparison voltage for the phase and the amplitude. The measured amplitude curve can, as proved by mathematical calculation, be replaced by a "two-circle-diagram" from which the azimuth, phase, and amplitude of the major axis of the oscillation ellipse can be determined. The ratio of the axes can be determined from the course of the phase with the aid of a simple mathematical relation.

According to geological investigation, the subsoil of one of the quadrants consisted of limestone deposited horizontally.

Eighty-four points of measurements distributed according to a polar coordinate scheme were selected here. The second quadrant was characterized by a tectonic disturbance of the "Kleperspalte," a water-conducting Keuper-graben in the midst of shell limestone. Forty-five points, distributed according to the rectangular coordinate scheme, were measured here.

The following results were obtained from measurements:

1. Lines of equal maximum amplitude for the electrical, as well as magnetic components of the field, are circles around the dipol; the circular form is influenced by the tectonic disturbances only if these disturbances are in connection with strong difference in conductivity.

2. The electric field of the dipol is linear only at the points situated at the prolongation of the dipol axis and its normals passing through the dipol; in all the other points the field is elliptically polarized. Lines of equal ellipticity are closed curves. Lines of equal ellipticity of the horizontal magnetic field are parabolas.

3. The electric current spreads out in the ground in the form of a wave extending from the dipol with a constant velocity of propagation, similar to the phenomenon of the transverse wave observed previously in a cable. The phase conditions in the main direction of the current are, owing to the ellipticity of the field, of no physical importance; lines of equal phases for fixed components, given by the coordinates of the dipol, were calculated and drawn; for the phase φ_z (vertical phase) and φ_y (component in the direction of the dipol-normal) the drawing resulted in circles around the dipol. From φ_y resulted a transverse wave twice as long as from φ_z.

4. Clear information on the tectonic of the subsoil could not be obtained from the results of the amplitude measurements and phase measurements carried out with probes, owing to a very strong influence of the heterogeneity of the surface of the soil.

5. From the geophysical viewpoint the elements of the magnetic field, a, φ_y, φ_z, could be evaluated the best, as for these elements the lines of connection of equal values in an undisturbed field were represented by concentric circles around the dipol; or in other words their value was proportional to the distance from the dipol.

6. Deposits of different conductivity cause disturbances of the normal course of the field, the phase curves reacting with a much greater sensitivity than the amplitude curves.--Author's abstract translated by W. Ayvazoglou.

(21) DIE GEOELEKTRISCHEN UNTERSUCHUNGEN MIT WECHSELSTROM

(GEOELECTRIC INVESTIGATIONS WITH ALTERNATING CURRENT)

By Wilhelm Geyger

Zeitschrift fuer Hochfrequenztechnik,
vol. 34, Nos. 5 and 6, 1930, pp. 184-190 and 228-233.

Contents of the article:

1. Introduction.

2. Theoretical bases of geoelectrical methods.

3. Principles of measurements used in geoelectrical methods.

4. Direct-current measurements.

5. Alternating-current measurements.

6. Representation of vibration phenomena that occur.

7. Method of the compensation measurement proceeding used.

8. Carrying out of terrane measurements in practice.

Theory and measurement proceedings of the electrical methods of prospecting are discussed, especially those carried out with alternating current of medium frequency of about 500 Hertz.

The author describes a complex "slide-wire alternating-current compensator (Schleifdraht-Wechselstromkompensator) developed by him, by which the determination of the two components of the alternating electromotive force perpendicular one to another (actual component and reactive component) can be determined directly.

Eighteen figures and 33 references complete the article.--W. Ayvazoglou.

(22) THE CONDITIONS FOR PROPAGATION OF ELECTROMAGNETIC WAVES IN THE EARTH ATMOSPHERE (IN RUSSIAN)

By S. Krutschkow

Journal of Applied Physics, Moscow, vol. 7, No. 3, 1930, pp. 61-79.

The results of investigations made by several authors show that the upper layers of the earth-atmosphere consist of oxygen, nitrogen, and helium. It seems that in the regions in which the aurora borealis occurs (altitude 80 - 130 kilometers) a certain part of the oxygen-molecules dissociates into atoms. Spectroscopic examination of the night illumination of the sky proved the existence of this dissociation.

After a discussion of various theories the author concludes that the temperature in the free atmosphere can not be established with certainty. The following are taken as possible temperatures: 180, 190, 200, 210, 220 and 300°, the most probable being those from 215 to 300°. Calculations made according to the formulas given in the article result in the conclusion that at altitudes of from 102 to 138 kilometers on, a complete dissociation of oxygen molecules takes place. This conclusion has been supported by the results of spectroscopic examinations.

Concentration of electrons calculated according to the formulas given

in the article reaches its maximum value (about 3.10^6 electrons per 1 $cm.^3$) at altitudes of from 113 to 168 kilometers. This concentration of electrons is not in contradiction with the results of investigations made on the propagation of electromagnetic waves.

The calculated altitudes of the Heaviside layer agree with the results of examinations. These altitudes depend on the length of the wave, initial angle of the wave, and the temperature of the stratosphere; they vary from 87 to 168 kilometers.--Author's abstract translated by W. Ayvazoglou.

(23) MESSUNGEN DER ANZAHL DER ATMOSPHÄRISCHEN ELEKTRIZITÄTSTRÄGER BEI NIEDERSCHLÄGEN

(MEASUREMENTS OF THE NUMBER OF ATMOSPHERICAL CARRIERS OF ELECTRICITY DURING PRECIPITATION

By K. Kähler

Gerlands Beiträge zur Geophysik, vol. 27, No. 2, 1930, pp. 226-240.

The number of atmospherical ions of all velocities, of small, intermedial, and large ions have been counted at the Potsdam observatory during rains, squalls, and thunderstorms. The results of these measurements are compared with synchronous registrations of the atmospheric potential gradient. The electrical process in rain clouds, especially the new Wilson-theory on the origin of the electrical charges of the drops, is discussed.--Author's abstract.

5. RADIOACTIVE METHODS

(24) LA TENEUR EN RADIUM DES EAUX PÉTROLIFIÈRES DE BAKOU ET DU DAGHESTAN

(RADIUM CONTENT OF PETROLIFEROUS WATERS IN THE REGIONS OF BAKU AND DAGHESTAN)

By B. Nikitin and L. Komleff

Comptes Rendus de l'Académie des Sciences, Paris, vol. 191, No. 7, 1930, pp. 325-326.

The authors studied the petroliferous waters in the regions of Baku and Daghestan (Caucasus).

Seventy-two samples taken from the region of Baku were examined. The emanation method was used. Measurements were made with Schmidt's electrometer. The highest content of radium, averaging 3×10^{-11} Ra per cent (in weight), was found in the waters of the Bibi-Eibat group (Baku), the richest at a depth of about 600 meters.

Twenty-two Daghestan samples were studied. The average content of Ra was 1.2×10^{-10} per cent. These waters contained about 0.03 per cent of barium. There was no proportionality between the contents of barium and radium.

Several wells are very rich in waters, their output being about 640,000 liters per day. Thus, the quantity of radium brought by them to the surface of the earth reaches about 0.2 gram per year.--W. Ayvazoglou.

(25) THE RADIOACTIVITY OF STONE MOUNTAIN

By James A. Hootman and W. S. Nelms

The Physical Review, vol. 35, No. 11, 1930, pp. 1431-1432.

This is an abstract of a paper presented at the American Physical Society held in Washington, D. C., April 24-26, 1930. The abstract reads as follows:

"The radioactive content of the waters which issue from the base of the unique geological formation known as Stone Mountain (Georgia) was determined. Tests were made of a number of large springs well distributed around the base of the mountain, and of other shallower sources above and below the 1,060-foot level. The method employed was the Schmidt shaking method, in which a known volume of water is thoroughly shaken with a known volume of air in a closed container. The resulting mixture of air and radium emanation is then pumped into an electroscope, and the rate of fall of the leaf is a measure of the radioactivity of the water. The electroscopes were calibrated by means of the Duane and Laborde formula, and corrections were made for temperature and pressure. This method has been shown by Ramsey to have an accuracy of about 3 per cent. A majority of the springs tested were highly radioactive, the value for the highest being 15,980 x 10^{-12} curies per litre. This value is nearly twice the maximum reported by Boltwood for 44 radioactive springs, and more than half as large as the maximum reported by Lester for 178 mineral waters of Colorado."--W. Ayvazoglou.

(26) THE NATURE OF COSMIC RADIATION

By L. F. Curtiss

The Physical Review, vol. 35, No. 11, 1930, p. 1433.

This paper was among those included in the program of the meeting of the American Physical Society held in Washington, D. C., April 24-26, 1930. The abstract of this paper, as published in the Physical Review, reads as follows:

"The Geiger-Müller tube counters placed vertically above each other show, in addition to accidentals, coincidences which have been ascribed to cosmic rays. Absorption experiments by Bothe and Kohlhörster indicate that these rays are corpuscular. The author has made observations with the poles of a large electromagnet between two counters, recording the number of coincidences with and without the magnetic field. With the field used of 7,000 gauss over an area 24 centimeters in diameter, a parallel beam of 10^9 volt electrons which passes through the upper counter would be deflected sufficiently just to miss the lower counter in spite of the fact that the Hρ for such electrons is approximately 3 x 10^6. Since the radiation is actually diffuse with a maximum in the vertical direction, using two counters one can expect only a slight

decrease in the coincidences if they are caused by high-speed electrons. However, if they are due to ultra γ-rays, no such effect should be observable. Making the theoretical allowance for accidentals, a decrease of the order of 25 per cent has been observed in the coincidences which may be attributed to cosmic radiation. This confirms tho existence of a corpuscular radiation of very high energy."--W. Ayvazoglou.

(27) UEBER DEN KLEINIONENGEHALT DER LUFT AUF HELGOLAND UND SEINE ABHÄNGIGKEIT VON METEOROLOGISCHEN FACTOREN

(ON THE CONTENT OF SMALL IONS IN THE AIR IN HELGOLAND AND ITS DEPENDENCE ON METEOROLOGICAL FACTORS)

By Oskar Mathias

Gerlands Beiträge zur Geophysik, vol. 27, No. 3-4, 1930, pp. 360-377.

Statistical investigations of the numbers of small ions obtained in Helgoland have revealed striking relations with meteorological and maritime factors, especially with those closely connected with the number of nuclei (direction of wind, transparency of atmosphere, sunshine.) The author recommends that in future in counting the ions the Aitken-nuclei be also counted at the time of observations as, according to investigations described in this article, they are of the greatest importance for the number of small ions. A direct influence of temperature could not be proved. Owing to small air-pressure changes during the time of observations (maximum 14 mm. Hg.), the investigation of its influence upon the number of ions was considered unnecessary and was not carried out. Interesting correlations with the degree of cloudiness and the tides were revealed. The increase in cloudiness caused an increase in the number of ions; this effect remained, of course, to be examined and explained. The average number of small ions was considerably greater at high tide than it was at low tide, a fact which seems to indicate a local formation of nuclei.

The relation of $\bar{n}_+ | \bar{n}_-$ showed a distinct correlation with the number of nuclei as well as with the relative humidity. This relation can be explained by the supposition that there exists a different capacity for accumulation of positive or negative small ions around the nuclei, respectively around the small water drops (formation of large ions). The relation $\bar{n}_+ | \bar{n}_-$ to the height of the waves is more complicated and requires further investigation.

The correction of the number of ions resulting from re-calibration of the anemometer does not meet the conclusions drawn by Hess concerning the balance of ionization. The relation between the number of ions and the average life of small ions shows a difference from the linear correlation for high values of $\bar{n}$; this can be completely explained by the deviation from the linear law of recombination.--Author's abstract translated by W. Ayvazoglou.

6. GEOTHERMAL METHODS

(28) THE STUDY OF EARTH TEMPERATURES IN OIL FIELDS ON ANTICLINAL STRUCTURE

By K. C. Heald

American Petroleum Institute, Bulletin 205, October, 1930, pp. 1-8

The purpose of the work described was, in general, to determine, if possible, the conditions responsible for variations in the rate at which temperature increases with depth, and for the changes that take place between one locality and another. More specifically it was proposed to determine, in so far as possible, relations between earth temperatures and the occurrence of oil and of helium-bearing gas in both local and regional geologic structure. It was further proposed to secure data on the relationships between earth temperatures and stratigraphic conditions, water circulations, and, in short, any geologic conditions that may be pertinent in connection with the discovery and the production of petroleum and natural gas.

The work has been centralized in areas where results will be most significant to the petroleum industry. Oklahoma, Texas, and California were chosen for study, since they not only offer favorable conditions, but also they are the areas where the results of the work seem best calculated to be of service to the industry. Work has been in progress in these three States since the spring of 1927 and the results obtained are briefly mentioned in this article.

Oklahoma. Local structure, chemical action, radioactivity, water circulation, effects of unconformities and effect of strong faulting are considered. Four maps are given.

Texas. Conditions of work, general accomplishment, stratigraphic conditions, relation of earth temperature to structure, regional metamorphism, and relation of rock temperature to oil fields are described.

California. Geologic conditions, plan of work, results, relation of temperature to circulating waters are examined.

Owing to the assistance given to the work described in this article by the U. S. Geological Survey and C. E. Van Orstrand, an outstanding American authority on earth temperatures and methods of measuring them, Heald considers the results obtained to be highly creditable.--W. Ayvazoglou.

(29) DESCRIPTION OF APPARATUS FOR THE MEASUREMENT OF TEMPERATURES IN DEEP WELLS; ALSO, SOME SUGGESTIONS IN REGARD TO THE OPERATION OF THE APPARATUS, AND METHODS OF REDUCTION AND VERIFICATION OF THE OBSERVATIONS.

By C. E. Van Orstrand

American Petroleum Institute, Bulletin 205, 1930, pp. 9-18.

In a brief introduction to the article Van Orstrand says that of the various methods that have been proposed for the measurement of temperature in

deep wells, two only, the electrical resistance thermometer method and the mercury maximum thermometer method, have yielded results of sufficient importance from the standpoint of efficiency and accuracy to justify their use in making an extended temperature survey.

These two methods are described in this article.

The advantages of the electrical resistance thermometer method over the mercury maximum thermometer method are, according to the author, so great that it will undoubtedly come into general use as soon as the main obstacles--the initial expense of the necessary equipment and the difficulty of constructing leads that will remain intact after being immersed in a mixture of crude oil and salt water--are overcome.

The results of the tests carried out by this method are shown in a depth-temperature curve of deep well, E. T. Price No. 9, South Penn Oil Co. From this figure the minute details of the temperature distribution are evidenced. A brief description of the apparatus is given.

The mercury maximum thermometer method has the disadvantage of being slow and tedious. Its chief advantages are minimum initial expense and the certainty of obtaining an accuracy of rather less than 0.3° F., regardless of the fluid contents of the well. The two methods of handling maximum thermometers, (1) by means of the bailer and (2) by means of a handoperated machine, are described. Thermometer holder and containers, as well as other pieces of apparatus, and several views of hand-operated machine are given in a series of pictures.

The second part of the article deals with some suggestions in regard to the operation of the temperature apparatus: (a) Tests with the oil well machinery, and (b) tests with a hand-operated machine. The operation is explained and the elimination of the chief sources of error in handling the thermometers is mentioned.

The last chapter of the article deals with some suggestions in regard to the correction and verification of the observed temperature data. The possibility of accurate location of the isogeothermal surfaces owing to the difficulty of obtaining a correct answer whether the well is in temperature equilibrium or not is discussed.

In conclusion the author makes statements in regard to the accuracy with which the isogeothermal surfaces can be determined; he says: "That they are smooth uniform surfaces before the drill penetrates the pay sands can not be denied. The real problem, therefore, is the location of these surfaces in their undisturbed condition. Barring the perturbations produced by flowing fluids previously discussed, there remains the instrumental errors of observation."

The errors in depth to the isothermal surface resulting from instrumental errors are estimated.--W. Ayvazoglou.

(30) DETERMINATION OF GEOTHERMAL GRADIENTS IN OIL FIELDS LOCATED ON ANTICLINAL STRUCTURES IN OKLAHOMA

By John A. McCutchin

American Petroleum Institute, Bulletin 205, October, 1930, pp. 19-61.

This report contains data which have been collected in the various fields and the conclusions which these data seem to warrant.

Curves from several wells are included and, where practical, are chosen from wells located near the centers of the fields and near the edges of the fields. The curves have been selected from wells which are considered to be in temperature equilibrium, and are the best available from these areas at the present time. It has not been deemed necessary to include curves from all wells tested; since many of them are very similar, and anyone who is further interested may plot the remaining curves from the data presented. The computed and tabulated data from 89 wells are also included.

While the data collected to date on this problem in Oklahoma are by no means sufficiently complete to permit other than preliminary conclusions to be drawn, as a whole, they do tend to support the general premise that the anticlines which have been productive of oil and gas have higher temperatures at given depths than the surrounding areas.

The variations within the individual fields are usually small, but measurable and uniform. The variations from area to area are quite large, and seem to bear a relation to the regional dip of the formations. This latter variation is so marked that all the data collected to date support it.

Due to the nature of the work, which involves the location and testing of the idle wells, most of the data have been collected near and within the fields proper, and very few tests have been made in dry holes located between the fields. This gives the problem a one-sided viewpoint, since it is impossible to know that geothermal anticlines do not exist where no true geological anticlines are to be found.

An interesting possible explanation is suggested by examining the available data. The anticlines which have been the most productive of fluid, either oil or water, show the largest and most uniform variations in temperatures at given depths. It is possible, then, that the removal of large quantities of water and oil from the higher points in the structures has allowed the warmer fluid from the greater depths, either down the dips of the formations or within the formations, to rise in the sands and occupy the pore spaces of the cooler rocks. It seems possible, then, that the abnormally high temperatures which are present in the rocks over the anticlines now may be due to this circulation. It is to be noted that the data which were collected in the Oklahoma City field, before an appreciable amount of fluid had been removed from the field, failed to show a marked variation in temperature with structure.

On the other hand, the data from the few wells located in the nonproductive areas indicate that these wells have rather markedly lower

temperatures at given depths as compared with the temperatures in wells located within the adjacent fields. In most instances, the data from the first 1,000 feet of the hole would have indicated an abnormally low temperature throughout the entire well.

Taken as a whole, the results obtained in Oklahoma to date on this project are neither consistent nor inconsistent enough to warrant any general conclusions as to the outcome of the project. The results do indicate that the problem is not a simple one, and that it will probably be solved by the collection of additional accurate data in the fields and dryhole areas, supplemented perhaps by some careful work on the heat conducting properties of sedimentary rocks in a well-equipped laboratory. Particular attention should be given to the collection of data from fields which are being developed, so that more information will be available from areas where there is no possibility that the operating conditions have influenced it.--Author's abstract.

(31) RESULTS OF DEEP WELL TEMPERATURE MEASUREMENTS IN TEXAS

By E. M. Hawtof

American Petroleum Institute, Bulletin 205,
October, 1930, pp. 62-108.

This paper presents a summary of the work of the writer carried out between the dates March, 1927, to January 1, 1929. During this period temperature measurements in deep wells were made in each of the oil yielding provinces of Texas, including the Gulf Coast salt dome province, the interior salt dome province of East Texas, the Luling-Mexia-Powell fault zone province, the Bend Arch, the Panhandle, the Permian Basin of West Texas and Southwest Texas.

A general picture of the geothermal conditions has been secured, which should not only justify certain preliminary conclusions, but should also permit effective planning and conduct of further and more detailed work in selected areas.

Equipment. The apparatus used in making temperature surveys was modeled after that described by Van Orstrand. Photographs of the apparatus are given.

Stem correction for maximum thermometer. All tests were corrected using the stem correction given by the U. S. Bureau of Standards. This correction may amount to as much as 1° F. in deep tests, although it is commonly much less.

Difficulty in securing suitable wells owing to their usually unsettled conditions is noticed.

Geological conditions dealt with. The wells in which temperatures were taken ranged in depth from less than 1,000 to more than 8,000 feet; thus a wide range of different structural types was covered by the work. Some idea of the regional structural conditions is given in a map of Texas showing by structure contours the writer's interpretation of the structural conditions,

the location of wells that yielded results of particular interest, and the number of feet of depth which, in each of the wells indicated, resulted in an increase in temperature of 1° F.

A series of maps, cross sections, and depth-temperature curves illustrate the article.

Original field notes and temperature tests carried out during this work, showing the county, field, company, well, name and number, are given in an appendix.--W. Ayvazoglou.

(32) GEOTHERMAL CONDITIONS IN OIL PRODUCING AREAS OF CALIFORNIA

By Anders J. Carlson

American Petroleum Institute,
Bulletin 205, October, 1930, pp. 109-139.

This report is a summary of the work concentrated mainly in the Santa Fe Springs and Long Beach fields of the Los Angeles Basin. Therefore, the results are particularly significant only in respect to relations between earth temperatures and local anticlinal structure of the Santa Fe Springs and Long Beach types.

The apparatus used was patterned after that designed by Van Orstrand with some changes made to facilitate operation of the equipment. A depth of 5,000 feet has been reached. Under favorable conditions a 4,000-foot survey could be completed in one day.

The reliability of measurements with this type of apparatus was demonstrated by the re-survey of some wells after a lapse of several months. The comparative average temperatures obtained by the author and by Van Orstrand for one well are shown in a table. This check made with different sets of thermometers is so close as to show that satisfactory results can be obtained by this method.

All tests were corrected using the stem correction formulated by the U. S. Bureau of Standards.

Factors affecting measurements are discussed.

In the attempt to determine whether or not a relationship exists between earth temperatures and geologic structure, the author has compared both isogeothermal surfaces and the distribution of reciprocal gradients to geologic structure. He concluded that in general the work in California indicated that the isothermal depth was a more reliable medium of comparison than the reciprocal gradient. This is explained by the fact that shallow temperatures seem to be more seriously affected by artificial conditions than do deep temperatures, and this would tend to affect the gradient of a well to a greater degree than it would the depth of the 100° isotherm.

Geological conditions under which the measurements were made and the types of wells used are described. Temperature determinations in 33 wells in

the Santa Fe Springs fields were made. The depth-temperature curves for these wells are shown in Figures 9 to 41. A table showing the geothermal data is given.

Geothermal data for 43 wells in Long Beach Oil field are shown in a table.

Conclusions from the study of the data presented, for both the Santa Fe Springs and Long Beach oil fields, are drawn.--W. Ayvazoglou.

(33) SULLA PREVISIONE DELLA TEMPERATURA NELL'INTERNO DELLE MONTAGNE

(PREDICTION OF TEMPERATURE INSIDE OF MOUNTAINS)

By Mario Bossolasco

Ergänzungchefte fuer Angewandte Geophysik,
vol. 1, No. 2, 1930, pp. 149-155.

After a few remarks on the practical value of different methods for the prediction of the temperature of rocks inside of mountains, the author mentions the importance of the position of the strata for the solution of these problems and explains by this the high maximum temperature which was established in the Simplon Tunnel.

He shows that the normal course of the geoisotherms can not be affected in a sensible way by hot springs. This is in contradiction to a confirmation made previously.--Author's abstract translated by W. Ayvazoglou.

7. UNCLASSIFIED METHODS

(34) MATHEMATICS AND THE PROBLEM OF ORE LOCATION

By Warren Weaver

The American Mathematical Monthly, vol. 37, No. 4, 1930, pp. 165-181.

The author divides his article into four parts:

A. Introduction. Some branches of earth physics by which the subsurface exploration is possible are considered.

B. The methods of geophysical exploration for ore and oil (magnetic method, electrical method, electromagnetic method, gravitational method, and the seismic method) are briefly discussed and illustrated by six figures.

C. The mathematical problems, as connected with the geophysical methods of prospecting are examined. Weaver notes the difference of the problems of geophysics from many classical problems which he characterizes as follows: First of all, they are two or three-dimensional problems; second, there are always three important regions of space involved, each with its own characteristic electrical properties--the air, the normal homogeneous earth, and the

ore masses; third, it is necessary to have exact knowledge near the sources of disturbance, rather than far away, as in the classical problem of radio signals; fourth, it is useful, in the electromagnetic case, to obtain solutions when the wave length is comparable with the dimensions of the diffracting ore mass, rather than small, as in the classical optical case; fifth, it is necessary, in general, to assign finite but different conductivities to the various regions under consideration; sixth, many regions of geometrical shapes, which were previously of only academic interest, now become of great practical interest; and, last, it is essential now that the solutions be usable from a computative point of view.

D. Conclusions. The first application of divining rods to the location of ore, as described in an old treatise of mining written by Georgius Agricola and published in 1556 (translated from the Latin by Herbert Hoover), is mentioned and the complicated equipment of the geophysicist of to-day with his dip needle, his voltmeter, radio apparatus, slide-rule and his textbook on partial differential equations is noted.--W. Ayvazoglou.

(35) GEOPHYSICS AND DEEP DRILLING SHAPE DESTINY OF GULF COAST

By Jack Logan

The Oil Weekly, vol. 59, No. 3, 1930, pp. 63-66.

The results of geophysical exploration and deeper drilling in the Gulf Coast field are summed up. According to information given by Logan, instead of approximately 40 domes and fields as in 1923, the Gulf Coast now has more than twice that number. The potentialities of the coastal area have besides not merely been doubled by the 100 per cent increase in the number of domes and fields, but they have been multiplied perhaps several times by accessibility in both old and new fields of deep sands which formerly could not be reached with the drill.

The immense increase in potentialities of the Gulf Coast as a source of oil is due to two principal factors: Deep drilling and geophysical exploration.

In this article the author discusses in more detail geophysical exploration as a factor by which the potentialities of the salt-dome oil country have been greatly widened.

Tables listing all old domes and geophysically discovered fields and prospects and a special map showing those numerous areas are given. The tables and the map reveal the brilliant record achieved by geophysics and the geophysicists.--W.Ayvazoglou.

(36) RECHERCHES SUR LES PERTURBATIONS ELECTROMAGNETIQUES, SISMIQUES ET SOLAIRES

(INVESTIGATIONS ON ELECTROMAGNETIC, SEISMIC AND SOLAR DISTURBANCES)

By Allert Nodon

Comptes Rendus de l'Academie des Sciences, Paris,
vol. 188, No. 10, 1929, pp. 725-726.

Investigations were made during almost two years by Boustos Navarette, Director of the Observatory del Salto in Santiago (Chile). Nodon's magnetograph was used.

Graphs obtained from observations made by magnetograph, seismograph, and galvanometer, as well as those made on the variations of sun spots proved the existence of a close relation between all of them.

From the results of these observations the important conclusion can be drawn that earthquakes can be predicted from the indications of the magnetograph several hours in advance.--W. Ayvazoglou.

(37) A BRIEF REVIEW OF ARTICLES APPEARING IN VOL. 2, NOS. 1, 2, AND 3 OF THE "BOLETIN DE LA ASOCIACION GEOFISICA DE MEXICO"

Boletin de la Asociacion Geofisica de Mexico,
vol. 2, Nos. 1, 2 and 3, 1930, pp. 1-78.

1. The physical principles of the gravitational method of prospecting, by B. P. Nikiforov. Continuation of the translation into English, pp. 15-22; into Spanish, pp. 39-45 and 61-66. Both parts translated by Prof. J. Korzujin.

2. Mathematical principles for topographic correction in gravimetrical work. By V. A. Olhovich. The article is to be continued.

3. Review of Geophysical Prospecting for Petroleum, 1929. This is Donald C. Barton's article published in the Bulletin of the American Association of Petroleum Geologists, vol. 14, No. 9, 1930, pp. 1105-1129 (see Geophys. Abs. No. 18), translated into Spanish by Manuel Alvarez.

4. Geophysical Methods of Prospecting, by Georges A. Boutry. Continuation of the French article translated into Spanish by L. Urguijo, Eng. (see Geophys. Abs. No. 8).--W. Ayvazoglou.

(38) SCHALLGESCHWINDIGKEIT UND TEMPERATUR IN DER STRATOSPHÄRE

(VELOCITY OF SOUND AND THE TEMPERATURE IN THE STRATOSPHERE)

By B. Gutenberg

Gerlands Beiträge zur Geophysik, vol. 27, No. 2, 1930, pp. 217-225.

The registrations of air waves confirm consequences drawn previously from ear observations. The results gained from the graphs show that the height in which velocity of sound and temperature begin to increase is not the same in all cases. Sometimes, especially in winter and in spring, it is as low as 20 kilometers. A simple method of calculation to find the sound velocity in the stratosphere is given.--Author's abstract.

8. GEOLOGY

(39) DIE RANDWERTAUFGABE DER GEODÄSIE

(THE BOUNDARY-VALUE PROBLEM OF GEODESY)

By F. Hopfner

Gerlands Beiträge zur Geophysik, vol. 27, No. 3-4, 1930, pp. 312-325.

The problem of determining the level extending into the sea-level (as it is known below continents it extends inside of the earth crust) by observed and properly reduced values of gravity comes under boundary-value problems of the second kind among which it holds the place between the exterior and interior space problems. In this article the author gives the solution of this problem with the aid of special functions of Green. The boundary-value problem of the first kind is solved also.--Author's abstract.

9. NEW BOOKS

(40) Krahmann, Rudolf. Die Anwendbarkeit der geophysikalischen Lagerstätten-untersuchungsverfahren, insbesondere der elektrischen und magnetischen Methoden (The Applicability of geophysical methods for prospecting deposits, especially of the electrical and magnetic methods). With 37 figures. Price, R.M. 2.50. This is the 3d volume of the "Abhandlungen zur praktischen Geologie und Bergwirtschaftslehre" issued by Prof. Dr. Georg Berg (Berlin, Geologische Landesanstalt).

(41) Stoner, Edmund C. Magnetism. E. P. Dutton and Co., New York, 1929. pp. Vii, 117, 20 figs., Price, $1.10. The topics treated are: The magnetic properties of atoms; diamagnetism as a general property of all matter; paramagnetism; ferromagnetism; the magnetic properties of the elements and the change of electrical conductivity in strong magnetic field.

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1 - The first figure refers to the number of the abstract; the second to the method of prospecting as indicated in the Table of Contents, and the third to the page.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING LAWS OF THE DOMINICAN REPUBLIC 1/

By I. Aitkens 2/

PREFATORY NOTE

This paper is one of a series of digests of foreign mining legislation and court decisions prepared in advance of a general report relative to the rights of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the laws of the Dominican Republic was prepared from the best available information in Washington, procured from the files of the Bureau of Foreign and Domestic Commerce and of the Department of State, and has been checked against the answers made by the American Minister, C. B. Curtis, to a questionnaire of the Bureau of Mines transmitted through the courtesy of the State Department. The assistance of the Commercial Laws Division of the Bureau of Foreign and Domestic Commerce in assembling material is gratefully acknowledged.

INTRODUCTION

The Mining Law of 1910, with amendments of 1914 and 1928, is at present in force in the Dominican Republic. The first and basic mining law of the Republic was passed in 1876, and although numerous changes and amendments have been made from time to time, the new laws and amendments thereto have not changed the original or basic law materially. However, these enactments have materially changed the legal methods governing the exploitation of the mineral resources of this country and have added new regulations and requirements to those concerning certain specific minerals, etc. The original or basic law still exists as it was first promulgated to protect the rights of the Dominican Republic itself, the rights of the landowner, and the rights of prospectors in general.

The Mining Law of 1876 remained in force, without amendment, until it was superseded by the law of May 25, 1904, an enactment modeled after the Mexican mining law and designed especially to encourage the investment of

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1/ The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from the U. S. Bureau of Mines Information Circular 6444."

2/ Rare metals and nonmetals division, U. S. Bureau of Mines.

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foreign capital in Dominican mines. The law of 1910 repealed previous mining legislation but was itself set aside temporarily by Executive Order No. 471, effective May 7, 1920. On June 3, 1925, however, the law of 1910, together with the amendments made thereto in 1914, was reenacted with certain additional minor modifications. This change was known as Executive Order No. 155. Further minor changes and amendments were made on May 1, 1928 (No. 915).

RIGHTS OF FOREIGNERS.

Any person, native or foreign, may register a claim and obtain from the Government the authorization to exploit one or more mines (Art. 8, Law of 1910).

OWNERSHIP OF MINERALS.

Generally speaking, mineral deposits belong to the State. The owner of the land has certain specified rights and privileges but his ownership is in the surface soil alone. He is, however, entitled to 2 per cent of the net produce of mines that may be exploited on his property (Art. 38, Law of 1910).

No one, not even the owner of the soil, has the right to open up or develop mines until a mining authorization or concession has been obtained from the Executive Power. However, placer mining for gold in the sands of the rivers and streams, without excavation, is not to be considered a mining exploitation. Such placer mining may be continued by the surface owner as long as the right to exploit these sands, etc., is not conceded to another, and, in fact, until another person actually exercises an acquired right by making necessary installations (Art. 1, Law of 1910).

The mineral substances for which a mining concession is necessary, are the following:3/

(1) Gold, platinum, radium, silver, mercury, iron, lead, copper, zinc, tin, antimony, nickel, cobalt, manganese, bismuth, arsenic, cadmium, chromite, beryl, molybdenum, selenium, tellurium, titanium, tungsten, uranium, vanadium, and zirconium (whether these substances are found native or in ore).
(2) Precious stones, rock salt, fossil resins, sulphur, asphalt, petroleum, oils, and coal.

Those substances for which a mining concession is not necessary, are the following:

(1) Mineral waters, building stone, clay of all classes, and sands that do not contain minerals for which a mining concession is necessary.
(2) In general, all other substances not specified as those for which a mining concession is needed.

3/ Article 2 of Mining Law of 1910, changed in accordance with amendments of March 25, 1925.

Any mineral deposit found in the surface soil upon land belonging to the Government may be exploited by any person who shall arrange with the Administrative Officer in charge of the Department of Public Works. It is the duty of that officer properly to protect the rights of the Government just as would be done if the deposit were the property of private parties or individuals. The Administrative Officer may dispose of such Government lands, for the purpose of exploiting such minerals, under terms the same as, or similar to, those governing the disposal of land for agricultural or other purposes.

According to the Civil Code, mines are real property; consequently a title to the concession rights shall be in writing. This writing or paper must be stamped by the President of the Republic, countersigned by the Secretary of Public Works, and registered by the proper enrolling officer of the Civil Government. Finally it shall be entered on the proper Government docket, for which service a fee of $1 is legally demanded.

PROSPECTING REGULATIONS 4/

Before prospecting on private lands, written authorization must be sought from the owner or from his legal representative. A petition to the Administrator of the Treasury is necessary in order to make investigations on State lands which are not already legally occupied by individuals. If the Administrator of the Treasury believes that such explorations or investigations might result in prejudices to the land, he may require a deposit of guaranty in accordance with Article 9 of the Mining Law, which says in substance:-

> In order to investigate mines in lands of another, it is necessary, after having obtained the permission of the owner, to obtain a special permit issued by the governor of the Province in which the mine is situated, which shall indicate the boundaries within which the investigation may be made. This permit may not cover a surface of more than 50 hectares.
>
> If the owner of the land refuses permission for such investigations, the prospector shall apply to the governor of the Province, who shall determine the guaranty that the prospector must deposit to cover all damages and prejudices that might result to the property. This guaranty must be determined by a judgment of experts previously named by the interested parties.
>
> These permits shall be copied in a special book kept by the Secretary of the Provincial Government, who shall charge for the copy of the permit a fee of $2 in gold.
>
> No permit shall be given for exploration in private buildings and their surroundings nor in places where major crops have been planted.

Six months is the usual term for exploration, counted from the date upon which the governor grants permission. This period may be extended for one equal term but only when the petitioner can prove that during the first term he performed investigations of considerable extent, and provided that he presents his petition for extension before the termination of the first period.

4/ Mining Regulations of July 27, 1910, promulgated by Ramon Caceres, President of the Republic.

During this period of exploration, the explorer alone shall have the right to present claims for mines within the explored zone.

MINING CONCESSIONS

Any person, native or foreign, has the right to denounce a claim, or solicit a mining concession from the Government. Such concessions which are granted by a resolution of the Executive Power and published in the official organ (Gaceta Official), convey the right to exploit only the mineral or minerals that were applied for (Art. 8, Law of 1910).

Petitions for one or more mining concessions shall be made in writing, in a paper suitably stamped and addressed to the Governor of the Province in which the mines are situated. Such petition shall state that a mine has been discovered on the desired property, describing concisely the exact location and class of mineral, its extent, and methods by which its extent has been determined (Art. 10).

This petition must be accompanied by a plot of the land showing the approximate area of the solicited concession, streams, and other topographical features included in the boundaries, the name of the site, the section, province, and, in short, all the data that an accurate description of the land requires (Art. 10).

The petitioner shall also furnish specimens in duplicate of all minerals found; and after exploitation has been begun, in the event that the mineral is found to be of a different nature from that described in the original application, the petitioner shall immediately make a declaration to that effect to the Executive Power (Art. 10).

The Governor registers the petition in a book kept for this purpose, in which notations are made of the surname of the petitioner and the day and hour of the presentation of the petition. Pending action the petitioner shall be given a certificate (Art. 11). Then, within seven days of such presentation, the Governor shall post up in customary places, notices of such petition with all information regarding intended exploitations. This notice shall also be inserted for the period of 30 days in daily and weekly newspapers of the province (or nearest province) in which the exploitation area is located (Art. 12).

Thirty days after the publication of the notices, provided all formalities have been complied with and no opposition has been offered, the Governor shall make a file of the case with the application and other documents, and within seven days following the last notice shall send the file to the Secretary of Public Works (Art. 12).

A duplicate of the whole mining survey shall be placed by the petitioner with the Department of Public Works within the 60 days following the beginning of operations. Failure to comply with this requirement is punishable with a fine of $25 in gold, plus an additional dollar in gold for each day of default after the 60-day period (Art. 21).

Mines must be exploited in a rational manner in accordance with the practice of good mining and in such manner as not to cause harm to third parties. In the case of water, the quantity used must be so regulated that no injury be done to other users, and unusual care must be taken in order to avoid poisoning such waters even when these are not used for drinking purposes (Art. 23).

The use of navigable rivers, and other means of transportation necessary for exploitation, is granted such operators in accordance with regulations governing public utilities. However, the proprietors are under obligation to construct wagon roads, railroads, and other means of transportation if such additional facilities are necessary for the successful conduct of their enterprise (Art. 24).

A right to construct telephone lines from mines to city offices is granted, but when such lines have been established, a report of their establishment must be made to the office of the Secretary of Public Works, and the Governor shall have use of these lines gratuitously (Art. 25).

REGULATIONS GOVERNING SURVEY STAKES, ETC.

The survey stakes that indicate a mining concession shall conform to the following conditions: 5/

(A).--They must be solid and constructed of cement, bricks, or wood of good quality, and shall be of a color and form distinct from neighboring landmarks.

(B).--They shall have markers engraved on copper plates with the name of the proprietor, the name that has been given to the mine, and a serial number separate and distinct for each mine.

Each proprietor shall have them registered with the Secretary of the Department of Public Works, who shall verify them to prove that there is no confusion with those already in existence, and who shall issue a suitable permit in consideration of a fixed fee of $5 in gold, plus the cost of the stamped paper on which it is executed.

They shall be kept in perfect condition by the concessionnaire during the existence of the concession.

(C).--These markers shall be located in such manner and with such numbers that it is clear from each one just what precedes and just what follows it.

All these requirements are deemed necessary in order that the proper mining survey may be filed in accordance with Article 14 of the Mining Regulations of July 27, 1910. These regulations require that all survey plans shall be drawn clearly and correctly, on strong paper or cloth, and their scale shall

5/ Art. 20 of Mining Law of 1910, as amended on March 25, 1925.

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always be in decimals of the metric system. Such plans shall contain in addition, the length of the sides in meters, the direction with reference to true meridian, the declination of the compass needle used, with the date upon which it was determined, and the area in hectares. Such plans shall be made by a competent surveyor, showing a complete topographical position of the concession area, with indications of rivers, and streams, and the exact points at which minerals were discovered.

ADVERSE CLAIMANTS

When in the judgment of the Governor a mining claim is concerned with zones already conceded, notice to that effect shall be given to the new claimant. If the claimant insists, the application must be registered with suitable reservation noted thereon. Such reservations must also be entered on the receipt given to him.6/

The concurring and opposing petitions shall be presented to the Governor and may be registered up to the last day of the 30-day period, which period is counted from date of first notice as provided by law. The law requires that these petitions and oppositions shall be registered in the book and that the interested parties shall be notified of all such oppositions (Art. 16). Such register shall be open to all persons desiring to see it, and whenever a petition is in controversy the Governor shall direct or advise the parties that recourse may be had before the common courts of justice, where the rights of all parties may be reviewed (Art. 17).

PROOF OF DISCOVERY

Law No. 915, (May 1, 1928) amends the Mining Regulations prescribed by the Executive Power on July 27, 1910, as follows:-

> Art. 26--Every applicant for a mining concession must present all facts that have led him to believe in the existence of the mine, which facts shall be examined by a commission of experts appointed by the Department of State for Public Works and Communications, - which commission must be convinced of the existence of the mine under consideration before granting a concession therefor.
>
> Par. 1.--From the decision of these experts, appeal may be made to the Executive Power for a determination of the case.
>
> Par. 2.--The expenses occasioned by the work of the experts shall be borne by the interested parties.

OBLIGATIONS OF PROPRIETORS OF THE MINES

Failure to begin mining operations within one year from the publication of a concession results in absolute forfeiture. Only in cases of unusual contingencies may an extension be granted by the Executive Power and such exten-

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6/ Changed in accordance with Art. 12 of Regulations of July 27, 1910.

sion shall be for one time only (Art. 37). However, where a railway is necessary for transportation of machinery, etc., a period of three years is granted within which to begin exploitation.

A deposit of $500 for a concession shall be exacted from each concessionnaire by the Secretary of Public Works, which sum shall be paid within the three-month period following the date of the concession application (Art. 36). This sum shall be paid to the Department of the Treasury.

For the following three reasons this sum may be returned:-

1. If underground inspections prove the impracticability of exploitations because of poverty of the mine.
2. If the mine is a vein mine; as soon as notice of the arrival of machinery to the value of $1,000 or more, is received by the custom inspector.
3. If the exploitation of metalliferous sands is to be done by hydraulic methods; as soon as 100 meters of canal or conducting pipes has been completed.

Each proprietor shall pay 2 per cent of the gross product of the mines to the public treasury; and in the case of mines situated on land owned by others (Art. 35), apparently an additional 2 per cent of the net output of the mines shall be given to the surface owners in return for the privilege of exploitation of their property (Art. 38).

PROHIBITIONS

Mining operations shall not be made at a distance of less than 50 meters from buildings, roads, railways, bridges, ditches, canals, watering places, and fountains, nor at a distance of less than 1,500 meters from fortifications. The regulations further require a distance of 500 meters or more from places designed as a deposit for inflammable materials, unless special permission has been obtained from the Secretary of State for Public Works, where buildings, etc., are on national property, and from the owners or their legal representatives when on private property.

EASEMENTS

Mining properties shall enjoy legal easements of right of way, aqueduct, drainage, ventilation, etc., but surface owners and also the owners of adjoining mining property are entitled to receive payment for any actual damage suffered. In the case of drainage or ventilation openings that benefit other operators, these other operators are liable to share the expense in proportion to the benefits they may receive therefrom (Art. 33).

The law requires that a concessionnaire shall indemnify the proprietor or proprietors for occupation of the land used, and for damages and prejudices occasioned in this connection. The amount of indemnity or damages to be made upon the estimates of experts and the amount for uncultivated land shall be a sum equal to double the value that it had before opening of such mining operations (Art. 29).

Appraisement of damages shall be made by the parties interested, and when no agreement can be made, they shall settle their differences in the manner prescribed by law (Art. 30).

EMPLOYMENT OF LABOR

The law requires that Dominican labor shall be used in preference to that of other nationalities. In case of a scarcity of such laborers, the concessionnaire is prohibited from importing laborers of the negro race (Art. 26).

TRANSFERS

A mining concession granted by the Executive Power gives perpetual ownership in the mine and is transferable like other property by gift, sale, or inheritance. However, no such transfer (Art. 3) except one resulting from succession, shall be valid without previous notice having been given to the Executive Official (Art. 5). Under no circumstances shall a transfer be made to foreign governments or states; nor shall such foreign government or states be accepted as partners. Any agreement with the afore-mentioned in addition to causing a forfeiture of the concession, is null and without any force whatsoever, regardless of any stipulations or provisions that may have been agreed upon (Art. 5).

FORFEITURE AND ABANDONMENT

Immediately upon the abandonment of a mine, the owner of the land is entitled to enter into possession. Regardless of the reason for this discontinuation of exploitation, whether because of completion of work or because of relinquishment, the owner may enter into immediate possession without the obligation of returning any moneys or fees that he may have received therefor (Art. 31).

However, when a concession is declared forfeited, the Secretary of Public Works places a marginal note to that effect upon the book in which such concession was previously registered, and the forfeiture shall be reported to the recorder in whose office the title was registered. A like notice shall be given to the Governor of the Province in which jurisdiction the mine is located.

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I. C. 6445

FEBRUARY, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

RESISTIVITY MEASUREMENTS UPON ARTIFICIAL BEDS

BY

JOEL H. SWARTZ

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I.C.6445.
February 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

RESISTIVITY MEASUREMENTS UPON ARTIFICIAL BEDS[1]

BY J. H. Swartz[2]

INTRODUCTION

During a series of resistivity measurements carried out in 1929 and 1930 a number of questions arose concerning the effect of topography on the character of the curves and the depths attained, the ratio of true depth to certain radii interpreted as representing depth, etc. To answer these questions it was decided to undertake a series of experimental investigations on artificial strata. It is the purpose of the present paper to present the results of the studies thus undertaken.

ACKNOWLEDGMENTS

The writer wishes to express his appreciation of a grant from the Rockefeller Fund of the University of North Carolina which permitted the construction of the artificial strata used in this study and the purchase of the materials required for them. Acknowledgment is also made of indebtedness to F. W. Lee and the U. S. Bureau of Mines for the use of direct-current instruments and apparatus, and to D. D. Carroll of the University of North Carolina for his kind permission to use his land for the construction of the hole and its contained artificial strata.

The writer wishes especially to express his thanks to W. T. Holland, of the University of North Carolina, for assistance without which the experiments could not have been completed, and to J. H. Watkins for invaluable aid. It is further a pleasant obligation to acknowledge the help given him in the construction of the experimental block by I. L. Martin, E. R. Scott, W. H. Hadley, and C. S. Maurice.

METHOD OF INVESTIGATION

A series of artificial beds was built up under conditions as nearly like those obtaining in actual field work as possible. In order to simulate such conditions accurately it was decided to place the beds in a large hole dug in the ground, thus doing away as far as possible with distortions of the electric field due to boundary conditions. It was further necessary to obtain a place free from pipes or other conductors and one where the walls of the hole would be or could be made impervious so that fluids placed in the artificial strata

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2 Senior geophysicist. U. S. Bureau of Mines.

could not leak out into the surrounding soil nor pass in that way from one bed to another. It was thus necessary to find a location where the subsoil was an impervious clay or where an ample supply of such clay was available. After an extended search an ideal locality was found where a diabase dike of Triassic age had weathered to a very impervious clay.

At this locality a hole 15 feet long, 12 feet wide, and 3 feet deep was dug. The hole was made large so that practically the whole of each potential hemisphere measured would lie within the hole in the artificial strata on which tests were being made. Sand from a nearby stream bed was used to form pervious beds for oil and salt water horizons. The impervious clay from the hole was used to seal off roots, animal burrows, and other openings in the walls and to form impervious beds separating the pervious strata from each other.

After the hole was dug it was divided into a series of 1-foot squares by a coordinate system, as shown in Figure 1, and the exact elevation of each intersection in the clay base was determined by transit. The beds described in greater detail below were then placed in the hole. The sand strata were formed by shoveling in the stream sand, which was then spread around and brought to exact level by packing, scraping, and filling, the final level being checked by transit.

Much difficulty was experienced with the clay layers separating the sands. The clay dried rapidly to a hard, brick-like mass which had to be broken up and moistened several hours before use. The amount of water used required rather careful adjustment since too much water made the clay too soft to retain its shape, whereas too little water left it too hard to permit working. Attempts were made at first simply to throw the clay in the hole and tamp it into place as lightly as possible. This procedure failed because it was either necessary to make the clay too soft to retain its shape and position or else to cause considerable damage and displacement in the underlying sands. It was finally found most satisfactory to take large chunks of the clay softened enough to be very slightly plastic while still retaining their toughness and tenacity and to chop them with a hatchet to the desired thickness and shape. The blocks so obtained were carefully fitted together and any spaces left filled by semiliquid or very soft clay. The top of each clay bed was carefully divided into 1-foot squares as before, and the exact levels of all intersections were determined by transit. Three sands were thus built up, separated and capped by layers of clay.

When the surface layer of clay had been added the hole was divided crosswise into two halves. In one half the surface was kept approximately level. In the other half a ridge-like mound 10 inches high was built up in the center. The shape of the surface is best seen on the topographic map, Figure 1.

The following is a list of the strata employed:

Clay 1. Base of hole, slanting upward to a central ridge. Elevations ranged from 0.00 at the north and south ends to +0.57 in the center (All elevations in hundredths of a foot above the lowest point of hole) See Figure 2.

Sand 1. This sand ranges from +0.45 at the edges to +0.82 in the center. It is divided into three parts. In the center an 18-inch wide lens of oil-saturated sand is separated by 3-inch walls of impervious clay from the wet sands on either side. On the north or "hilly" side of the hole this sand is filled with salt water carrying 30 pounds of

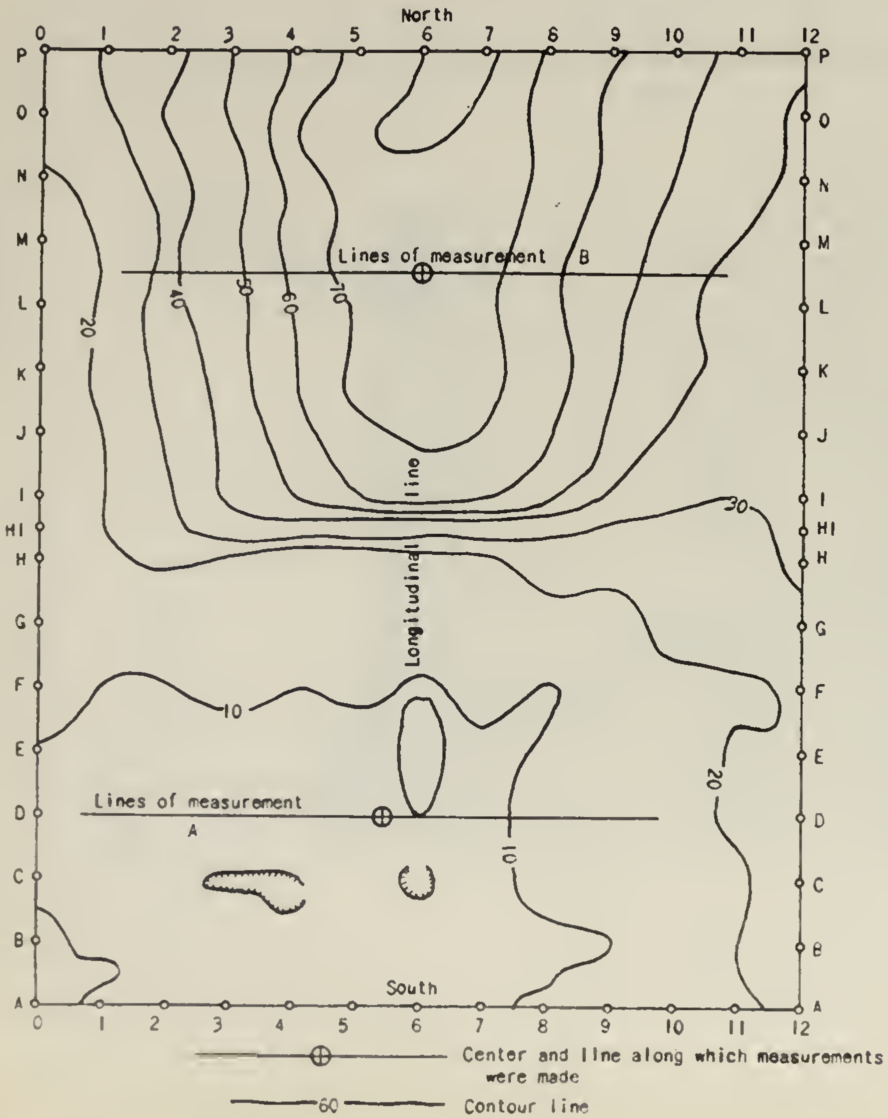

Figure 1.- Topographic map of surface of completed experimental block (top of Clay 4)

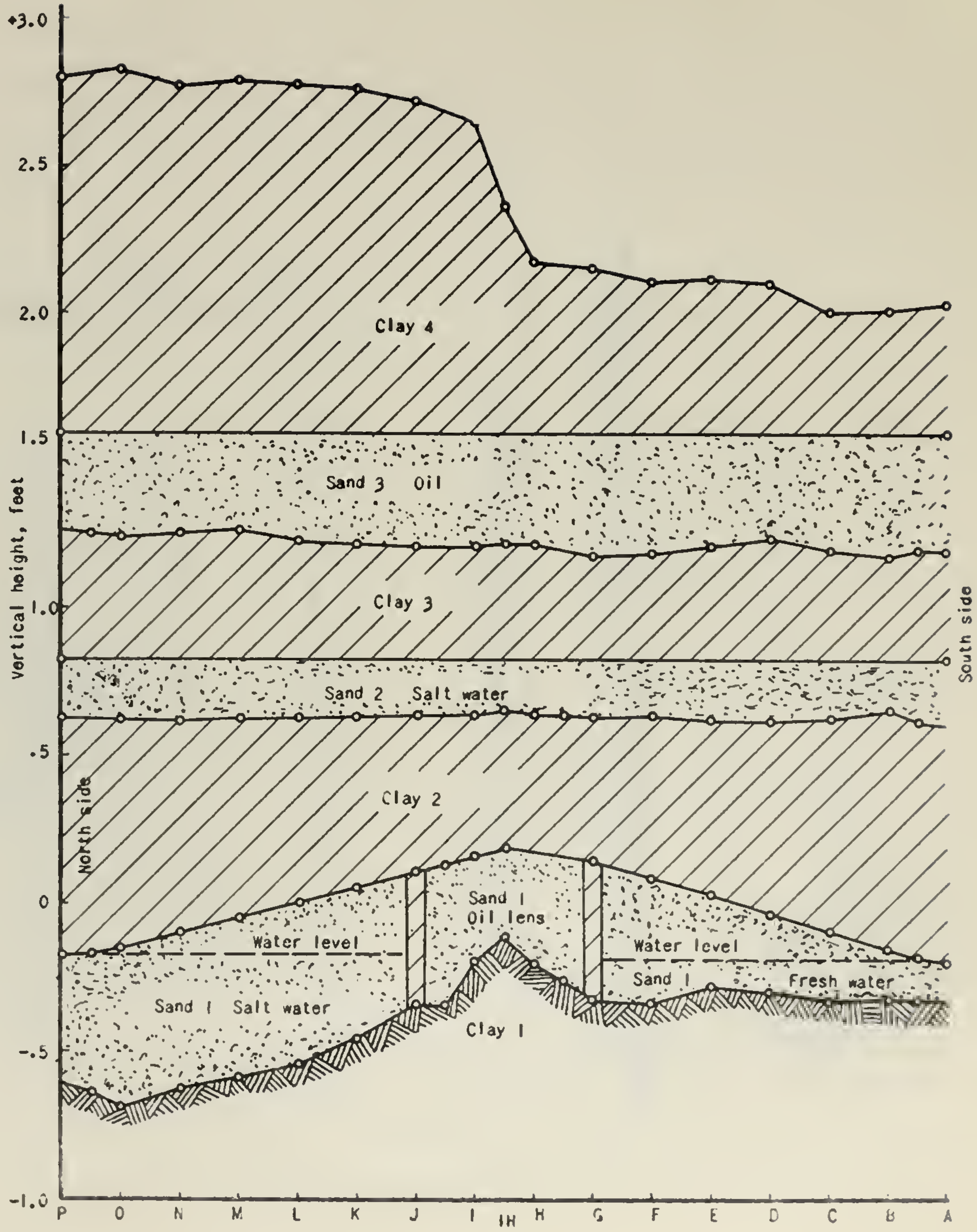

Figure 2.- Medium longitudinal section through experimental block

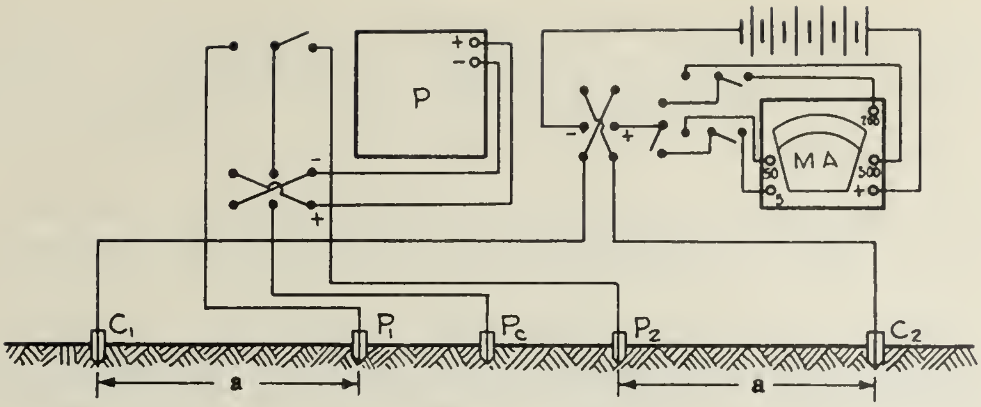

Figure 3. – Connections for Lee's method. C_1 and C_2, current stakes; P_1, P_2, and P_c, potential pickups; P, potentiometer; M A, the milliammeter

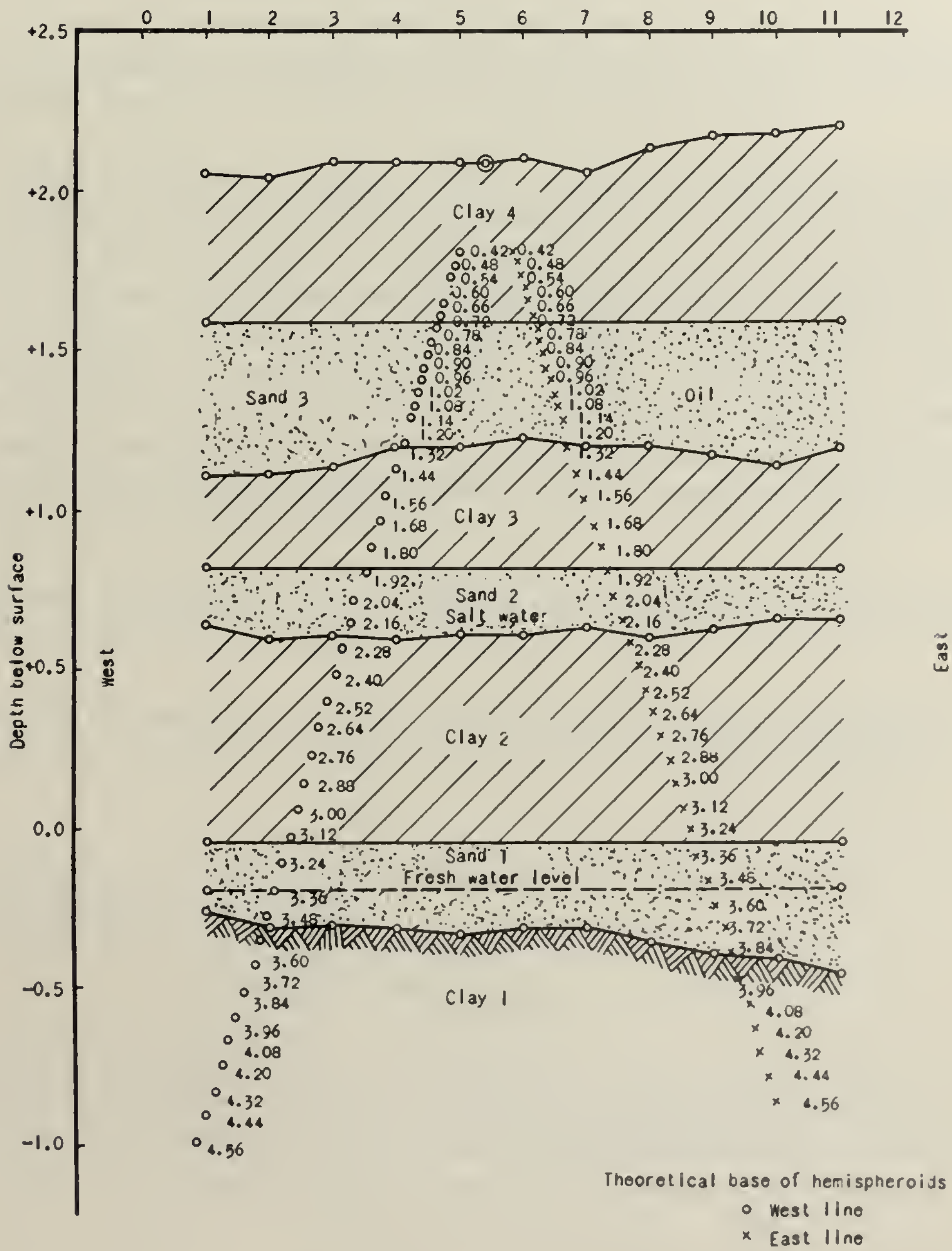

Figure 4.- Vertical section through the flat side of the experimental block along line of measurement A

salt dissolved in 150 gallons of water and filling the sand to a depth of +0.47 foot. On the south or "flat" side the sand is filled with fresh water to a height of +0.45 foot. Above these two levels in either half the sand is dry; See Figure 2.

Clay 2. The upper surface of Clay 2 is horizontal and stands at an average level of +1.29 feet above the lowest point in the hole.

Sand 2. This, the first entirely horizontal sand, has a thickness of 0.20 foot, its upper surface at a level of +1.49 feet above zero (the lowest point of Clay 1, the clay base). This sand is filled throughout with salt water formed by dissolving 60 pounds of salt in 160 gallons of water.

Clay 3. Horizontal layer with its upper surface at an average elevation of +1.89 feet.

Sand 3. Oil sand. Horizontal layer of thickness 0.37 foot, with upper surface at elevation of +2.26 feet. Filled with 160 gallons of Bunker C oil, a heavy, black fuel oil with a gravity of about 14° B.

Clay 4. Final clay cap. Upper surface shown on topographic map, Figure 1. Elevation of lowest point of surface, which serves as zero point in Figure 1, is +2.67 feet. This clay had to be put on during a heavy rain, in order to protect as much as possible the underlying oil sand. As a result a considerable amount of oil oozed up into it, in some places through it, while being laid. These oil seeps were later blocked by adding clay and surface soil after as much of the oil-soaked clay as possible had been removed. A certain residuum of unremovable oil remains to cause some irregularities in the resistivities of the lower half of this clay, as shown by the curves.

The arrangement and position of the beds is shown in Figure 2, which gives a median, longitudinal section through both hole and strata.

Measurements on the beds were made by Lee's method of partitioning, Wenner's method and the single electrode probe method, using direct current with potentiometer and milliammeter.

Figures 4 and 10 give profiles through the beds along the lines in both hilly and flat sides on which measurements were made.

RESULTS OF MEASUREMENTS

Direct Current. Direct current was furnished by one to three radio B batteries. In view of the diminutive character of the beds and the correspondingly small intervals, 0.08 foot, between measurements the ordinary porous cup nonpolarizable electrodes were too large to be satisfactory. Electrodes which proved entirely satisfactory were made from ordinary porcelain tube insulators by closing the bottom with a little retort cement. A length of copper wire was placed inside, the whole filled with a concentrated solution of copper sulphate with an excess of copper sulphate crystals, and the top closed with a small

cork. Because of the density of the porcelain it was found necessary to let the electrodes stand for a couple of weeks before use, to allow the walls to become saturated with the solution. For current electrodes, brass bolts 5/16 inch in diameter were used. The results of the individual experiments follow.

Lee's Method of Partitioning

The details of Lee's method have been adequately described elsewhere[3] and need not be repeated here. Figure 3 shows the electrical connections used. In Figures 5A and 5B are given the results obtained along line A in the application of this method to the southern or flat half of the experimental beds. The figures beside each point on the curve give the distances of the current stakes from the center stake for those points. Two curves are given, the 5A curve representing the line measured to the east of the center, the 5B curve that to the west.

We discuss first the eastern curve. Due to the dryness of the surface the curve starts at a comparatively high resistivity, 14,000 ohms per centimeter cube. This is followed by a rapid drop as the ideal hemispheres penetrate into deeper and more moist clay. This continues with minor fluctuations to a depth of 0.44 feet (current distance 0.66 feet). Beyond this point the curve rises rather rapidly to a maximum between points 1.20 and 1.32, though nearer the latter figure as shown by the shape of the curve. Turning to the diagram of the beds (fig. 4) we note that the base of Clay 4 is at a depth of 0.50 feet below surface. This at first sight appears to be slightly deeper than that indicated by the curve. As has already been pointed out, however, weather conditions made it necessary to cover the underlying oil sand so rapidly that a considerable amount of oil was caught in the lower part of Clay 4. The presence of this oil is clearly indicated by the fact that the rise toward the maximum begins at the shallower depth.

In Figure 4 the symbols (circles and cross marks) represent the theoretical position of the base of the ideal equipotential hemispheres measured for each position of the current stake, the distance of the current stake from the center for that particular point being given by the figures beside the symbol. Since these figures also identify the corresponding points on the curves they form a more convenient distinguishing mark than the depth and will hereafter be used for identification purposes instead of the depth.

The base of the oil sand in Figure 4 is exactly where indicated by the curve in Figure 5A, between the points 1.20 and 1.32 but much closer to 1.32. The sharp drop to point 1.44 is probably due to a small amount of water at the base of the oil sand. Below point 1.80 there is a slight increase in the steepness of the curve followed by a minimum at or near point 2.16. This coincides accurately with salt-water Sand 2 whose top, as seen in Figure 4, lies between points 1.80 and 1.92 and whose base lies just below point 2.16. Following a slight rise to 2.28 there is a rapid drop through moist clay to point 3.24, the base of Clay 2. Below this point lies Sand 1. As pointed out, this sand is filled with fresh water, but only to the level indicated in the diagram. Above this point the sand is dry, or relatively so, and hence has a higher resistivity. This is clearly shown by the increased resistivity at points 3.36 and 3.48 in the curve, Figure 5A. These two points are

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3 - Lee, F. W., and Swartz, J. H., Resistivity Measurements on Oil-Bearing Beds: Tech. Paper 483, Bureau of Mines, 1930, 12 pp.

Swartz, J. H., Oil Prospecting in Kentucky by Resistivity Methods: U. S. Bureau of Mines Technical Paper, to be published.

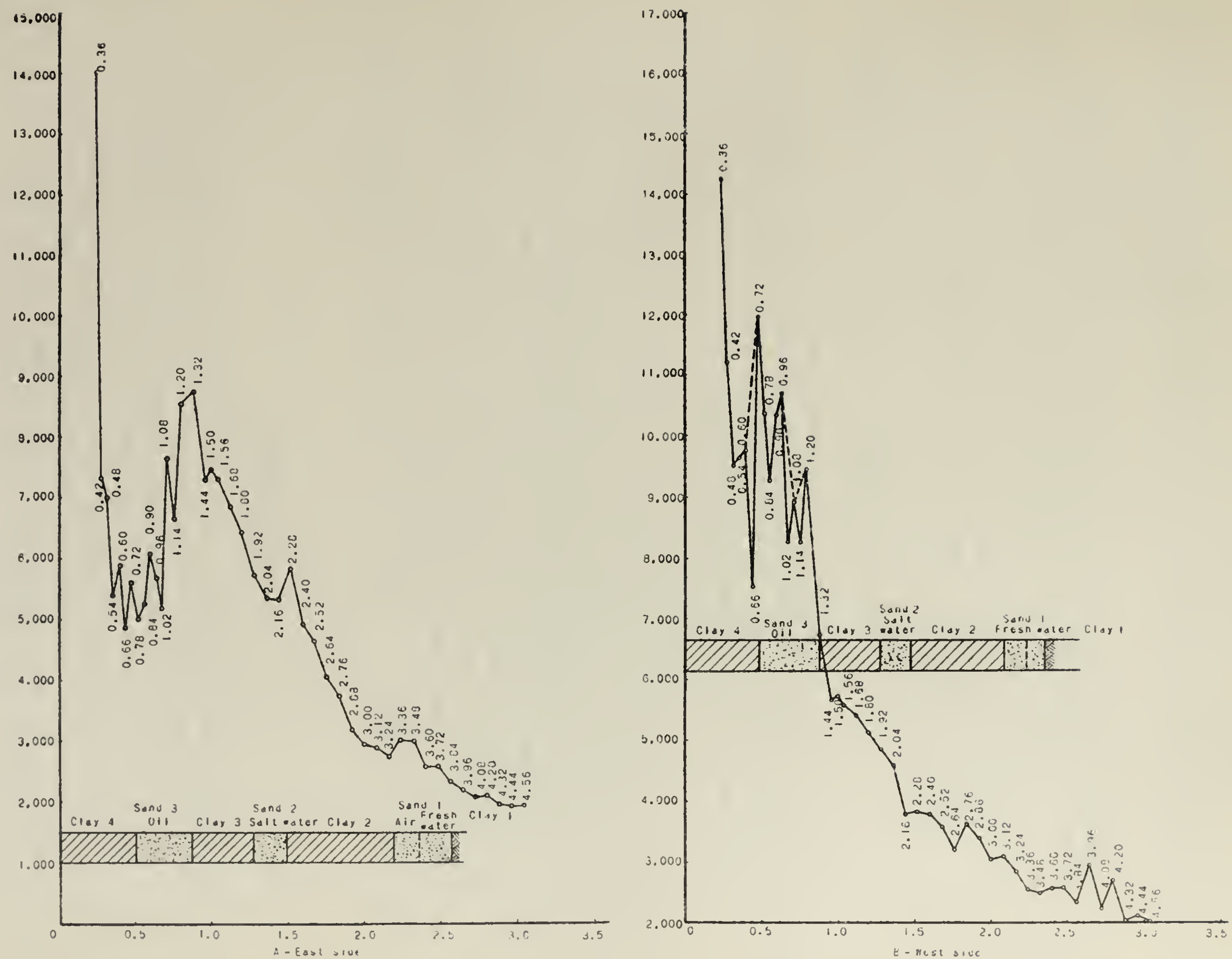
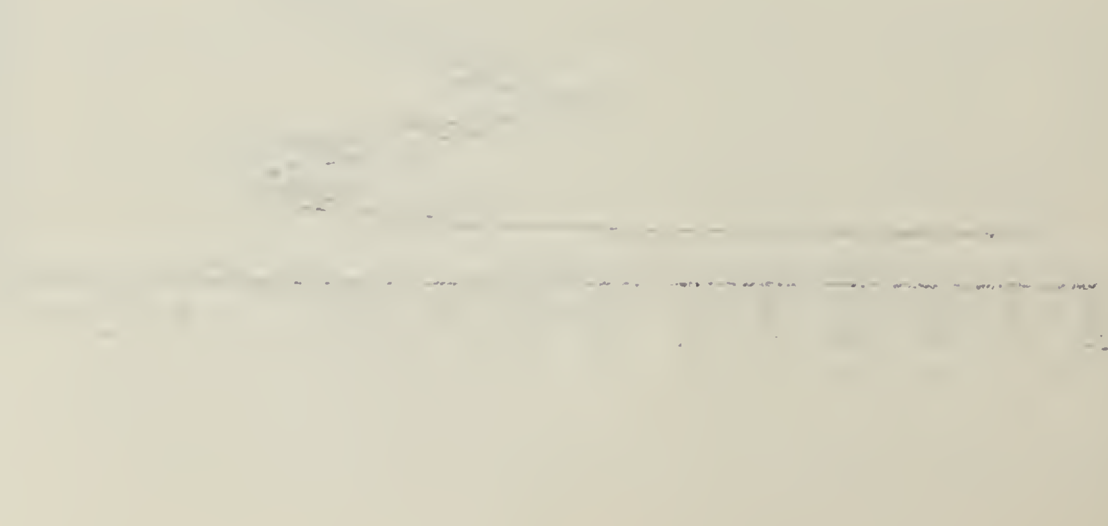


Figure 5.- Curves obtained on flat side of experimental block by Lee's method of partitioning



exactly those shown in Figure 4 to be located in this upper dry portion of the sand. This is followed by a steady decline through the fresh-water filled portion of the sand and the underlying moist clay to 4.32, where the curve flattens out. The exact correspondence between the theoretical depths as indicated by the curve and the actual depths as measured in building the strata is striking.

Much the same characteristics are found in the west curve, Figure 5B. The break here is at 1.20 and is sharper, due very probably to a larger amount of water at the base of the sand on this side caused (a) by the greater slope to the west of the center and (b) by the fact that the east side of the oil sand was covered first, leaving the west side of the sand exposed to a heavy downpour of rain for a much longer time. The effect of the salt-water is first seen at 2.04 instead of 1.92. As shown in Figure 4, however, a displacement of only 0.01 foot would suffice to throw the 1.92 point up into the clay. Thus a slight drop in Clay 3, due to tamping, or a slight soaking of the salt water into the underlying Clay 2 would more than account for this apparent disparity.

In this curve also points 3.36 and 3.48 represent a minimum rather than a maximum. It is to be noted, however, that points 3.36 and 3.48 on this side, due to topographical differences at the surface, corrected for in locating these points in Figure 4, lie not in the dry part of the sand but below in that portion occupied by the fresh water.

Three points stand out from this study of the curves: (1) The insulating effect of the oil and short circuiting effect of the salt water do not prevent the registering of resistivity variations in the underlying beds (for instance, the "high" due to dry sand at 3.36 and 3.48 (fig. 5a) on the east line). (2) The depth at which a given resistivity change occurs is given with striking accuracy by plotting the apparent resistivity against _a_ as here done (where _a_ is one-third the distance between the current stakes and is calculated from the formula = 383 a E/I, in which I is the total current flowing in the circuit, and E is the potential drop between the pick-up electrode and the central electrode). (3) The depth reached must always be measured _beneath the current stake_, not beneath the center. This, of course, is due to the fact that the equipotential hemisphere on which the measurement is made surrounds the current stake, not the center. This factor will be further discussed at a later point.

Wenner's Method

If the central electrode in Figure 3 is disconnected and the potential drop measured between P and P we have the connections and method first suggested by Wenner. Figure 6 gives the results of a series of measurements made by this method about the center and along the line used for Lee's method above. While the curve shows the same general characteristics, the individual features are less pronounced. Thus the top of the oil sand, point 0.66, is much less clearly indicated, though the base of the sand, point 1.20, is still clearly drawn. The salt water sand at 2.16 is indistinct. The dry upper portion of Sand 1 does not show. The inferiority of this method when compared with Lee's method of partitioning is due to the fact that lateral changes in topography and to lateral variations in the stratigraphic character and structural position of the beds cause the two halves of the electric field to differ, often strikingly. In Lee's method each half is measured separately. In Wenner's method the two halves are averaged together and their individual characteristics thus concealed.

Single-Electrode Probe

A series of measurements were made at the same center using the method of the single-electrode probe, connections for which are shown in Figure 7. Two procedures were employed. In the first a series of concentric but overlapping hemispherical shells of regularly varying thickness were measured, the outer radius, a, being increased by a constant interval, in this case 0.08 foot, but always kept equal to twice the inner radius, as first suggested by Lee. In the second a series of concentric but contiguous, nonoverlapping hemispherical shells was measured, the shells having a constant thickness of 0.08 foot, both radii being increased by 0.08 foot between measurements.

Figure 8 gives the results of measurements made by the first method, that of overlapping shells of regularly varying thickness in which a always equals 2b. The interpretation of this curve is difficult. The rapid rise shown by the initial part of the curve would the rise begins is too shallow, even after correcting for the oil trapped in the base of Clay 4. The peak at a in Figure 8 can not represent the base of the oil sand, since the curve continues to rise to the point c; and there is no doubt, as shown by the other curves, that it should drop steadily after reaching the moist Clay 3. The maximum at point c would thus appear at first sight to mark the base of the oil sand. However, a comparison with Figure 5B, measured along the same line, indicates that peak e is perhaps even more likely to represent that point. In either case, however, the plotted depth is somewhat too great instead of too shallow.

There seems no doubt that the minimum, h, represents the base of the salt water Sand 2. If so, this constitutes a still greater discrepancy between plotted and true depth, the plotted depth being again too great. Thus the plotted depth is too shallow for the first part and two deep for the latter part of the curve. When one part fits, the other does not. The factor by which a, the outer radius of the shell, must be multiplied to obtain the true depth, is thus not a constant but a variable. Although it appears to decrease in value with depth, the data is insufficient to permit conclusions concerning the regularity or significance of this change.

Lee has pointed out that it appeared necessary to multiply the outer radius of the shell by 1.4 to obtain the correct depth. A similar factor appears in the charts given by Crosby and Leonardon . In both these cases a simple arrangement of two contrasting strata was involved. In the present experiment the stratal configuration is more complex, eight essential stratigraphic units being involved, resulting in a greater variation of the correction factor. It thus appears that with increasing stratigraphic complexity the equipotential hemispheres in the method of the single electrode probe are subject to such considerable and variable distortion that there remains no constant relationship between plotted and true depths. Since the relationship between the curve and the strata involved is a variable one, it becomes difficult to interpret it satisfactorily.

Shells of Constant Thickness

Figure 9 gives the curve resulting from the second procedure, the measurement of a

4 - Lee, F. W., Joyce, J. W., and Boyer, Phil, Some Earth Resistivity Measurements: Information Circular 6171, Bureau of Mines, 1929, 16 pp.

5 - Leonardon, E. G., and Kelly, S. F., Some Applications of Potential Methods to Structural Studies: Am. Inst. Min. and Met. Eng., Tech. Pub. 115, 1929, 18 pp.

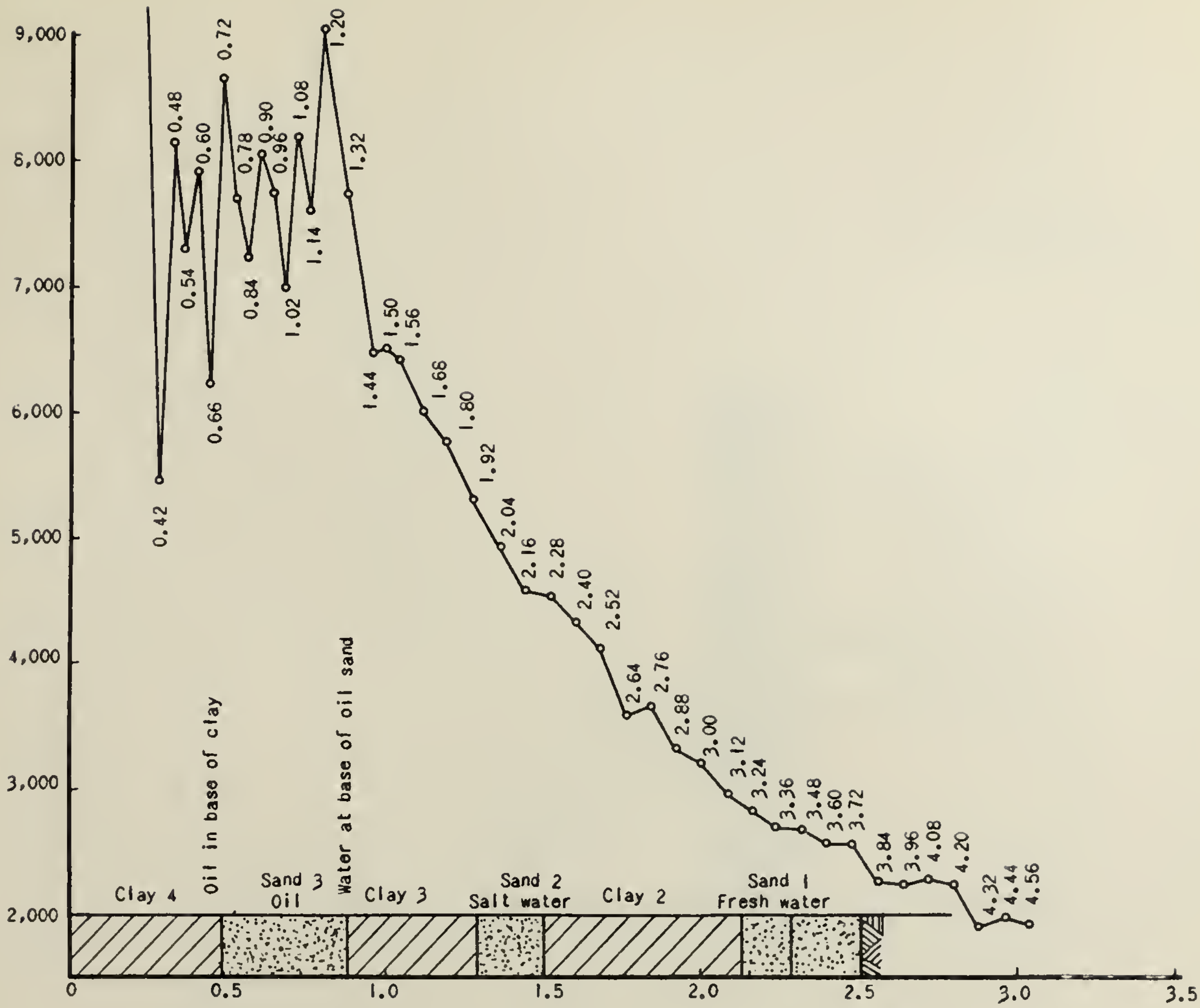


Figure 6.– Curve obtained on flat side of experimental block by Wenner's method

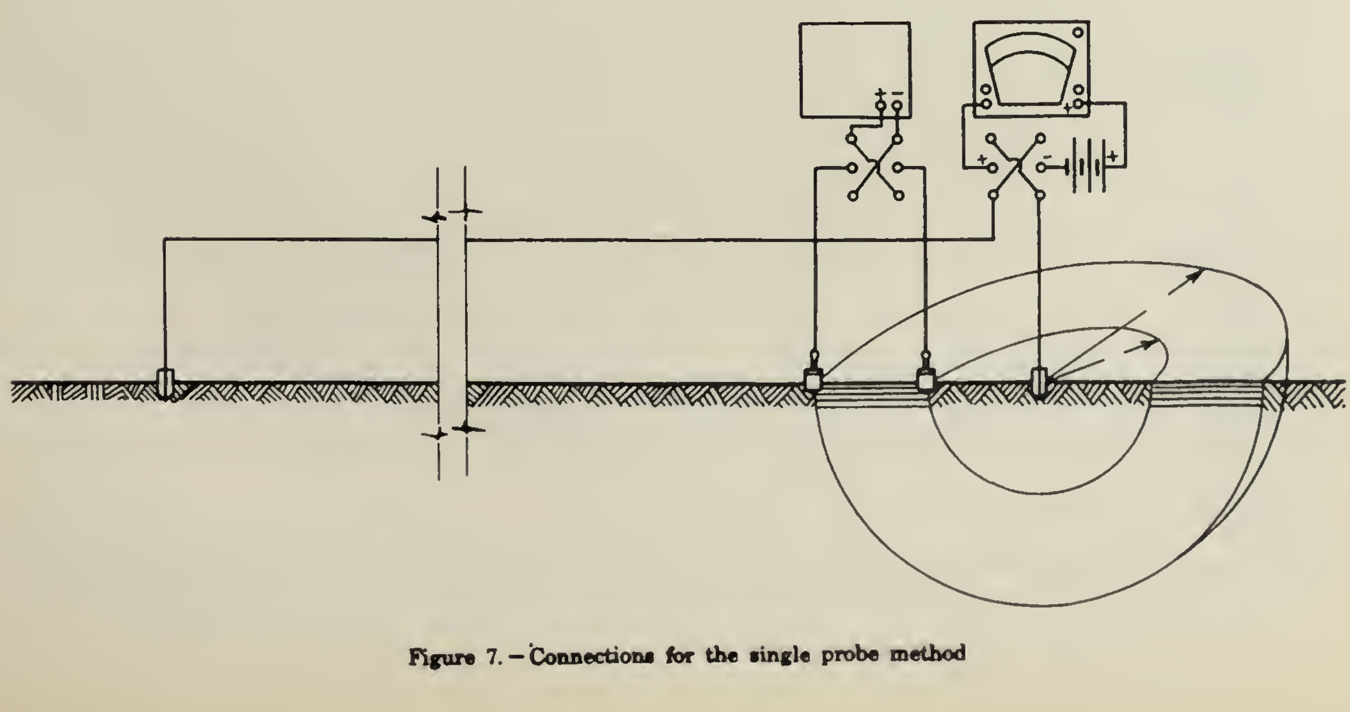


Figure 7. – Connections for the single probe method

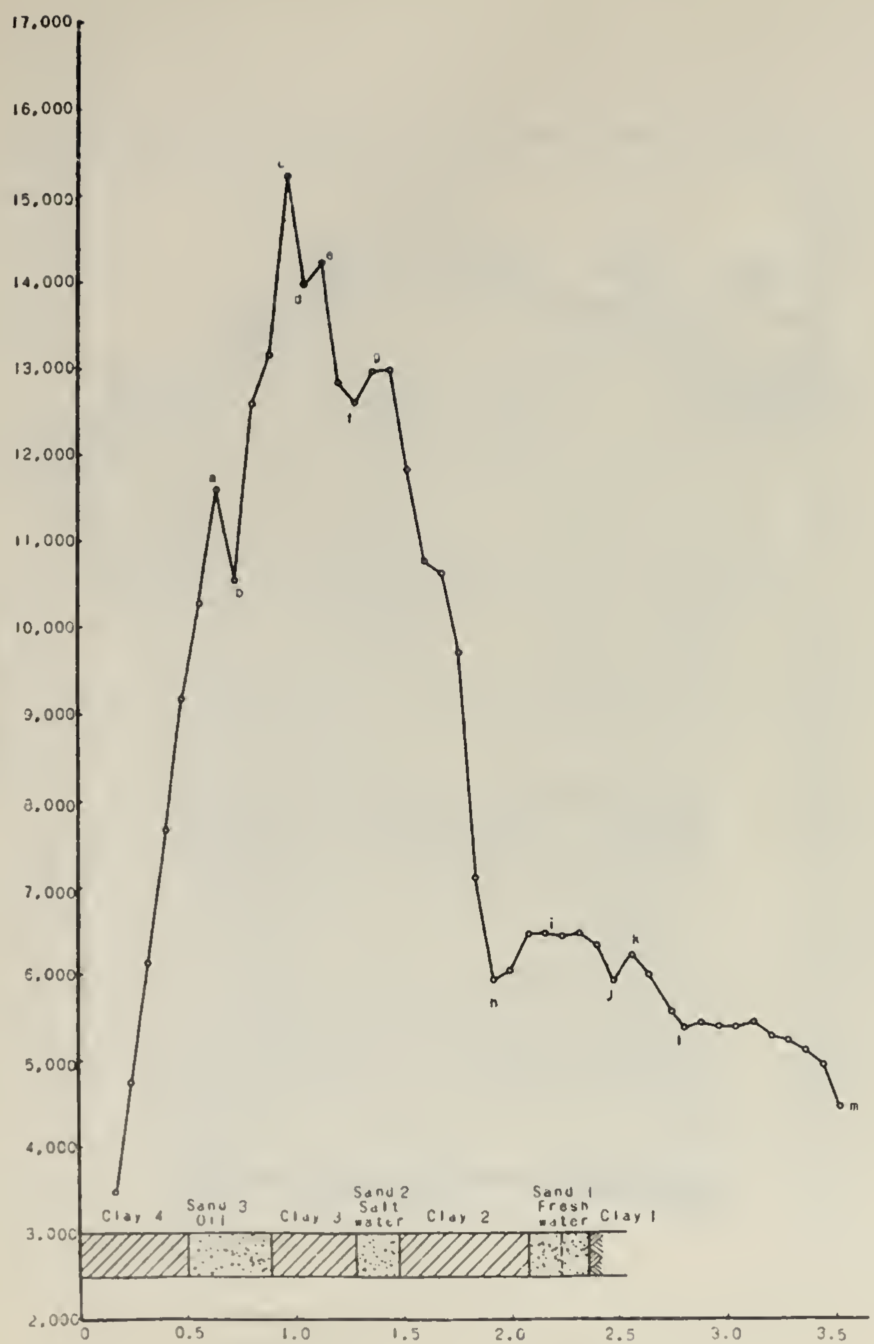


Figure 8.- Curve obtained on flat side of experimental block by the single-electrode probe, keeping a = 2b, as suggested by Lée

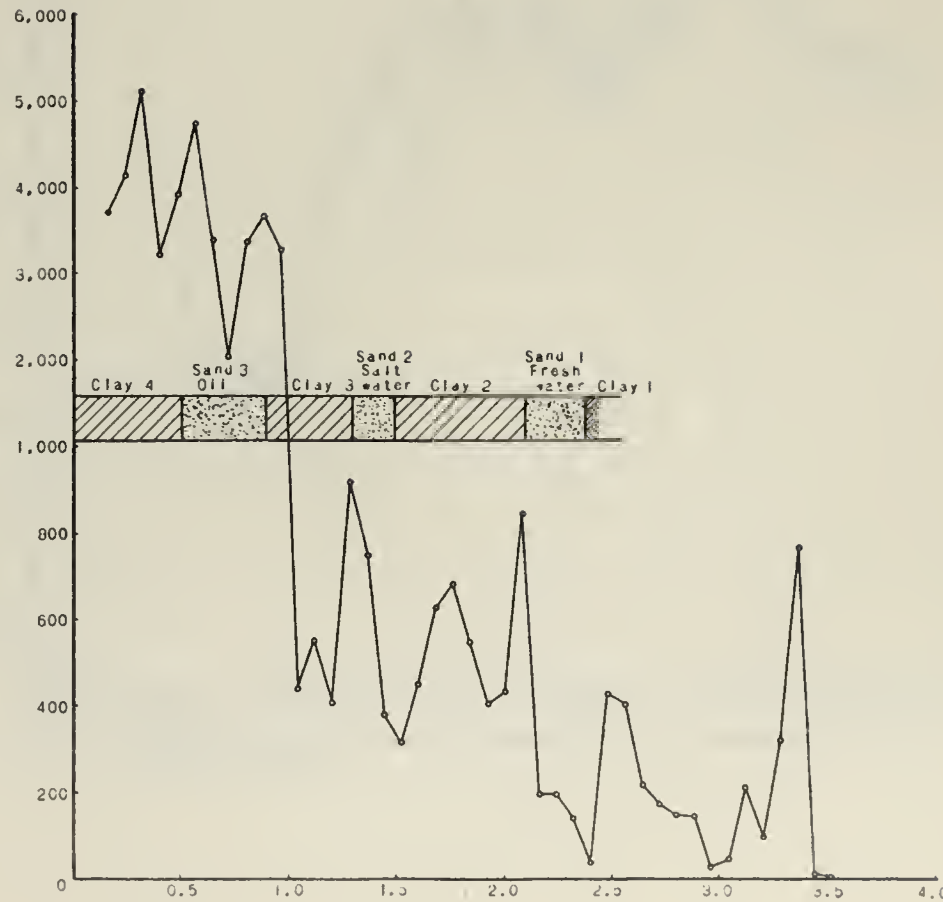


Figure 9.- Curve obtained on flat side of experimental block by the single-electrode probe, using shells of constant thickness, following Schlumberger

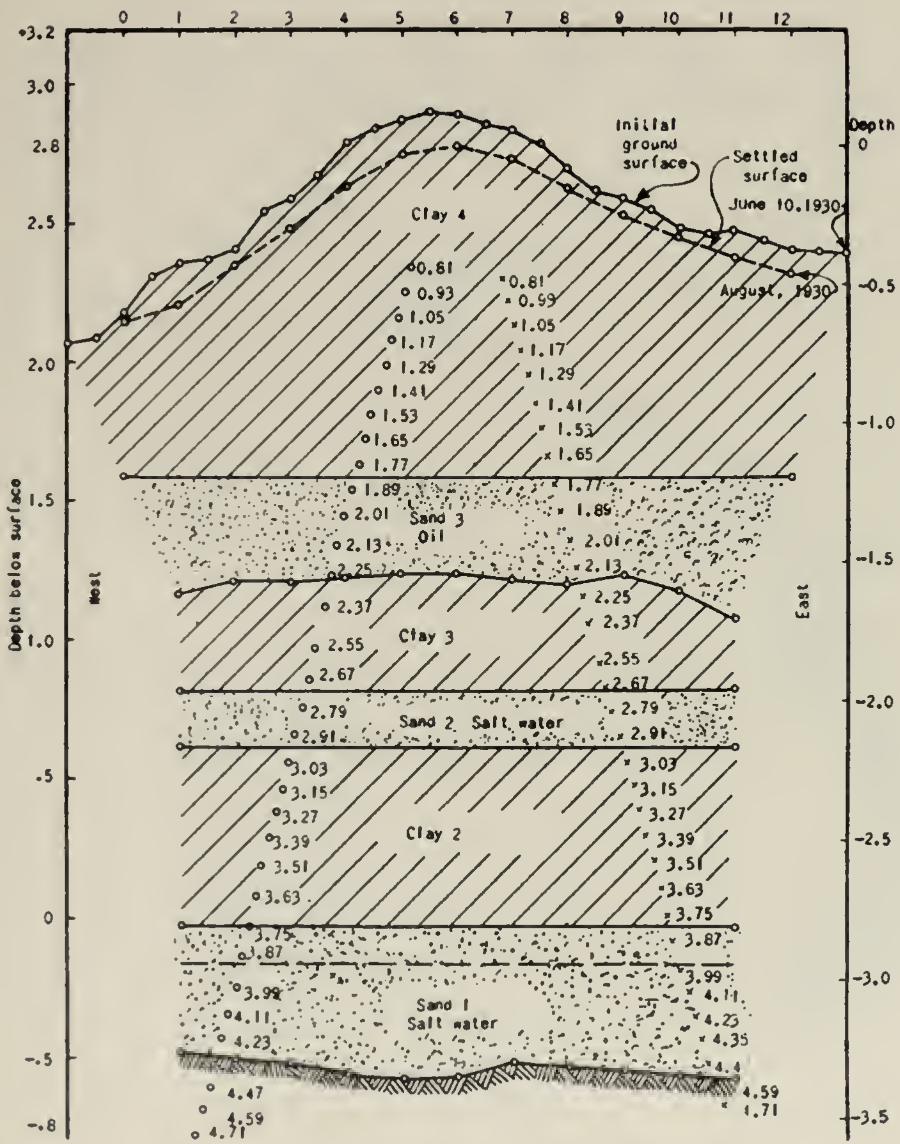


Figure 10.- Vertical section through hilly side of experimental block along line of measurement B

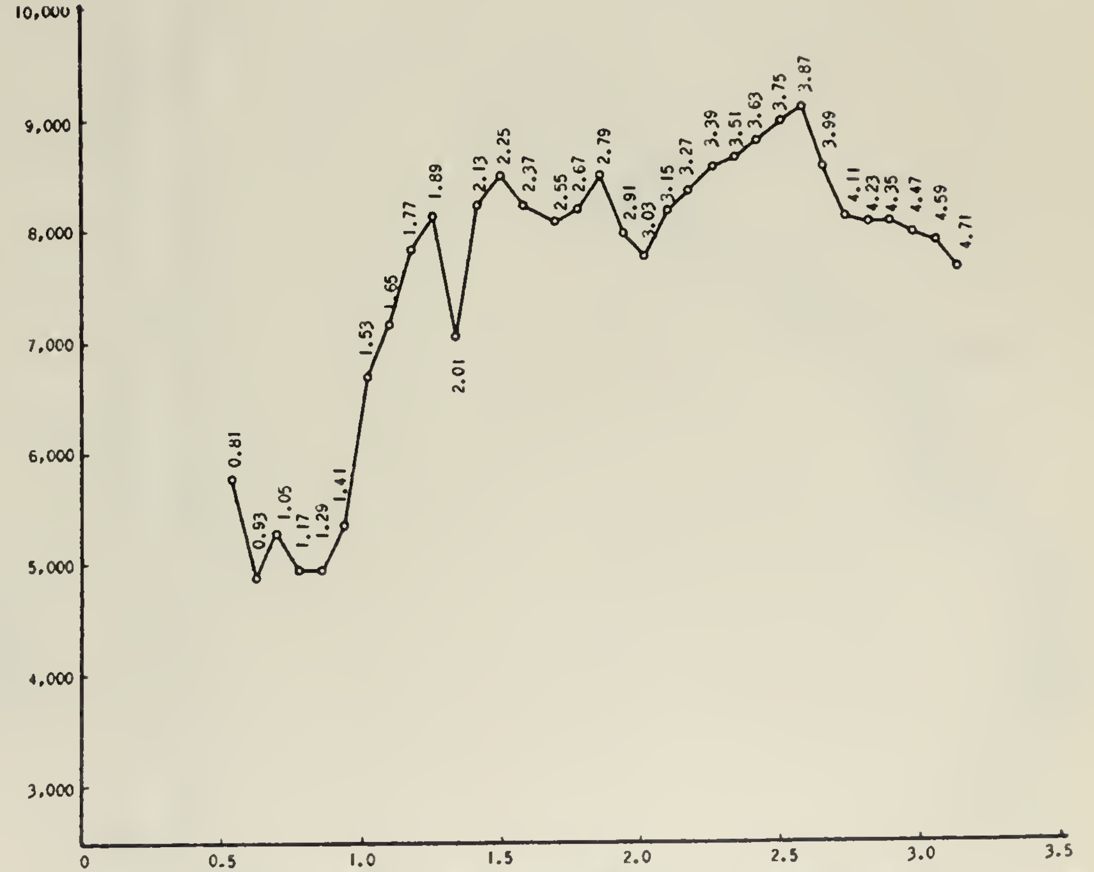


Figure 11.- Curves obtained on the hilly side of the experimental block by Lee's method of partitioning with direct current. West line

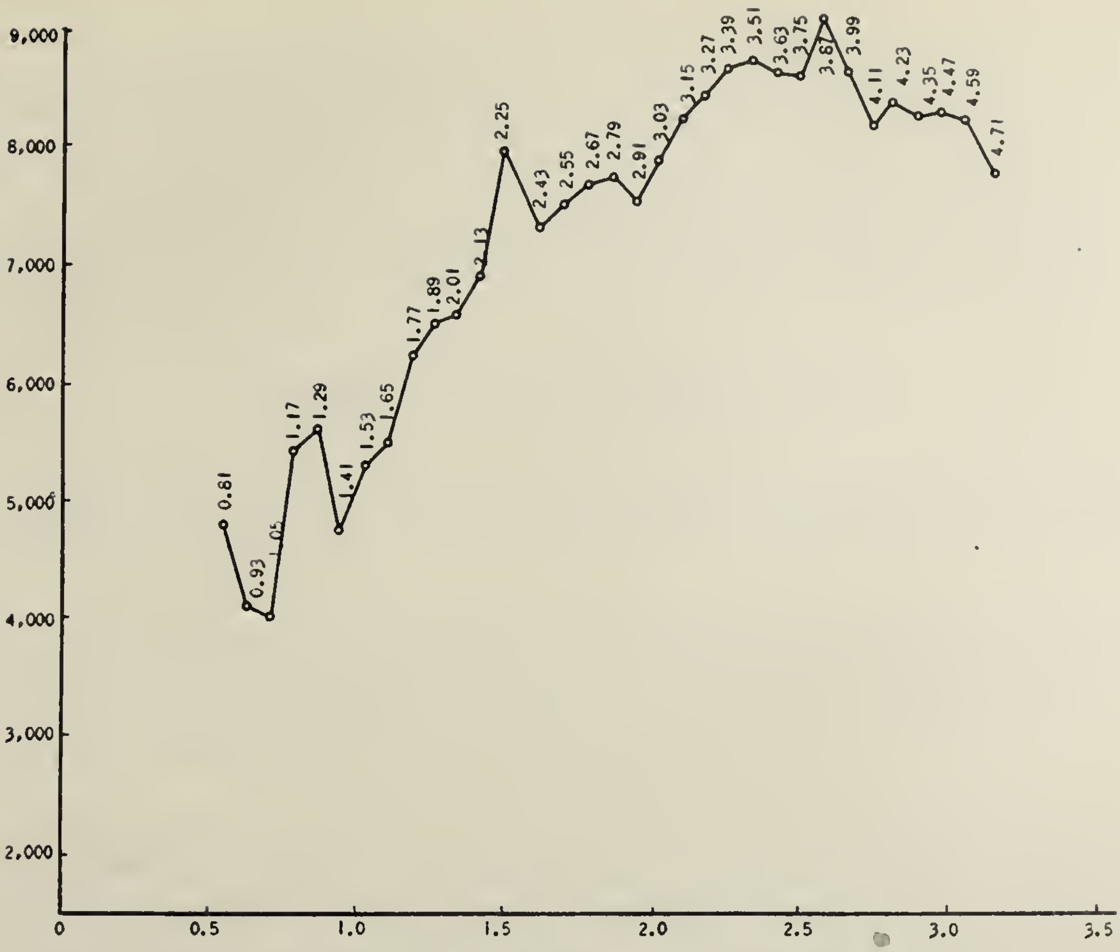


Figure 12.- Curves obtained on the hilly side of the experimental block by Lee's method of partitioning with direct current. East line

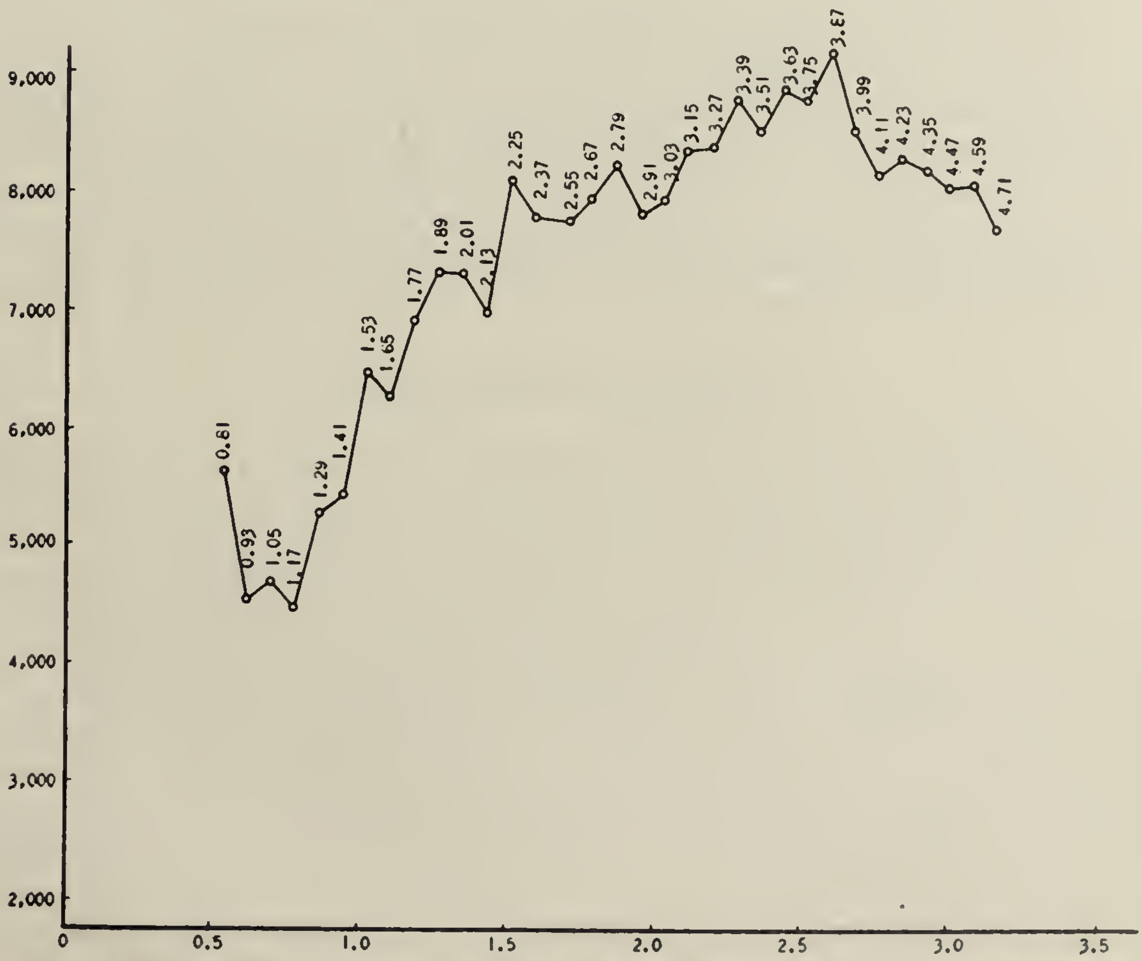


Figure 13.- Curves obtained on the hilly side of the experimental block by Wenner's method with direct current

series of contiguous, nonoverlapping shells of constant thickness, in this case 0.08 foot. The rapid variations and marked irregularity of the curve is its most marked feature. There is no apparent resemblance between this curve and that of Figure 8, nor is there much basis for interpretation. It is obvious from the curve that irregularities in surface resistivities are so great as to mask variations due to subsurface conditions. This experiment confirms the view long held by Lee that data obtained by the use of shells of constant thickness is too greatly affected by surface variations in resistivity to have any value in the interpretation of subsurface conditions.

MEASUREMENTS ON THE HILLY SIDE OF THE BEDS

The measurements discussed were all made on the flat side of the experimental beds. The remaining discussion will deal with measurements made on the hilly half of the experimental block, as shown in Figure 1.

Figure 10 gives a section through the artificial beds along the east-west line used in all the following experiments. The upper, solid line gives the surface of the beds as it appeared during the Lee and Wenner measurements. The lower, broken line gives the surface as it appeared during the single probe measurements.

As already noted, topographic variations need to be considered in interpreting resistivity measurements, especially in determining the depths attained. The chief purpose of the following experiments on the hilly half of the block was, first, to confirm more completely the conclusion that a topographic correction is required and, second, to determine its nature.

In view of the apparent exactness with which the value of $\underline{a} = 1/3\ C_1C_2$, (fig. 3) expresses the depth attained, as noted in the foregoing experiments with Lee's method of partitioning, it was assumed as a first approximation that the proper topographic correction would be obtained by plotting below each current stake position the corresponding value of $\underline{a}$. The points indicated by circles and cross marks in Figure 10 were obtained in this way, the numbers beside them again representing the distance of the corresponding current stake from the center. The correctness of this assumption stands or falls by the accuracy with which the points so located fit the curves obtained here by Lee's method, as discussed subsequently

Lee's Method of Partitioning

The curves obtained by the use of Lee's method on the hilly half are shown in Figures 11 and 12. The west line shown by the curve in Figure 13, reaches a maximum at the 2.25 point which corresponds exactly to the base of the oil sand as shown in Figure 10. There follows a drop through the moist Clay 3, with a rise to a second peak at 2.79, followed by a drop through the salt water of Sand 2 to a minimum at 3.03. From this point on it rises rapidly to a second peak at 3.87 in the dry upper portion just above the salt water in Sand 1. This is followed by a fall through the salt water, with a flattening out at 4.11, followed by a renewed drop when the clay base is reached at 4.35. The curve fits the points as plotted in Figure 10 with surprising accuracy. The only discrepancy, a slight one, is the apparent downward shift of the salt water in Sand 2. This is more probably due to a downward soaking of the salt water into the upper part of Clay 2, which had been considerably dried out and cracked before the sand was placed in it. In this way the salt water might easily have been lowered the 0.06 foot required. This would leave a thin dry zone at the top of the sand, thus accounting for the maximum at 2.79, and also would give raise to a saturated zone

at the top of the underlying clay, thus accounting for the minimum at 3.03.

The east line, in Figure 12, rises also to a maximum at 2.25, indicating the base of the oil sand. This is followed by a drop into the moist Clay 3, whence the curve again rises to a maximum at 2.79, followed by a drop into the salt water minimum at 2.91. This time point 3.03 is definitely below the salt water zone. Clay 2 persists in its effect to 3.75, where we again get a rise to a maximum at 3.87 in the dry upper portion of Sand 1. This is followed by a drop into salt water and a flattening out at 4.11, as in the west curve. The break at 4.59 possibly indicates passage down into the basal clay. All points fit satisfactorily, with the possible exception of the oil maximum at 2.25. This point is apparently offset 0.05 foot.

The most important fact to be noted here is the accuracy with which the points of the curve coincide with the points plotted in Figure 10. It follows that the assumption on which they were based is probably correct. A topographic correction is thus necessary where the current electrodes are moved during measurements. The depth attained is further equal to <u>a</u> (= 1/3 C C) and should always be measured from the surface of the ground downward <u>beneath the corresponding current stake position</u>. This experiment indicates that the stratum reached is that lying at that depth at that point.

To refer all measurements to a fixed point, such as the center about which measurements are made, it is necessary to subtract algebraically from the depth, <u>a</u>, the difference in elevation between the fixed point and the current stake position, calling all higher elevations positive and all lower elevations negative. To determine the stratum penetrated it is also necessary to correct for the dip and strike of the beds between the two points.

<u>Wenner Method</u>

Figure 13 gives the results of measurements with Wenner's method at the same place and along the same line. Since the two curves obtained by the method of partitioning are so much alike, indicating a close similarity in the two halves of the field, it is not surprising that the Wenner curve is unusually clear-cut and resembles them closely. Even under these favorable conditions, however, it is less regular and clear-cut than the partitioning curves.

This method is subject, of course, to the same topographic corrections as Lee's method, with the added difficulty, however, that different corrections in the two halves of the field will frequently throw the corresponding hemispheres into different strata, thus still further confusing the curves. These errors for deep observations may become a critical factor.

<u>Single-Electrode Probe</u>

The curve in Figure 14 was obtained on the hilly half of the beds by the method of the single electrode probe, using overlapping shells in which <u>a</u> always equals 2<u>b</u>, utilizing the same line (line B, fig. 1) and center. The curve again presents difficulties of interpretation because of the variable relationship between plotted and true depth. The minimum at a depth of 0.40 foot is perhaps approximately at the base of Clay 4. The maximum at 0.72, or that perhaps between 0.96 and 1.04, appears to represent the base of the oil Sand 3. The minimum at 1.52 feet probably represents the base of the salt water in Sand 2. Contrary to

the results of the curve shown in Figure 8, the plotted depth is here always too shallow and must always be multiplied by a factor greater than 1.0 to obtain the true depth. Again, however, the correction factor is not a constant, but a variable which appears to decrease with depth.

CONCLUSIONS

Of the various methods tried, Lee's method of partitioning gives the sharpest, most clear-cut, and most easily interpreted results. Wenner's method fails because it averages together the two usually quite dissimilar halves of the field. Lee's method of partitioning has the further advantage that it gives a simple depth relationship, which is a constant one. The depth to the stratum displaying the resistivity shown by a given point on the curve is in these experiments always equal to one-third the distance separating the current stakes for that point, as measured beneath the particular current stake position for that point. This relation does not appear to vary with changes in the resistivities of the beds penetrated - a rather surprising result since it was to be expected that the equipotential hemispheres would be very differently distorted by beds of different resistivities, with corresponding variations in the depth recorded.

In general the single-electrode probe gives less clear-cut results than the method of partitioning and has the further disadvantage of a variable ratio between the outer radius, a, and the true depth.

The necessity for a topographic correction where current stakes are moved during measurements, as in the Lee and Wenner methods, is one of the important results of the present study. Since the hemispheres center about the current stakes and not about the center of measurement, measurements of depth must be made below the current stake. To refer all measurements to a fixed point, such as the center, differences of elevation between the fixed point and the current stake must be subtracted algebraically from the depth below the current stake, designating as "negative" all points below the center, or other fixed point, and those above as "positive." To correlate the stratum reached with those below the fixed point it is necessary to correct for the difference of elevation of the stratum between the fixed point and current stake. This may be easily calculated, if the strike and dip of the beds are known, from the formula

$$h = D \cos \beta \sin \delta,$$

where h is the difference in elevation desired, D is the distance between fixed point and current stake, δ is the angle of dip, and β is the azimuth angle between the dip-bearing and the bearing of the line joining current stake and fixed point. Nomographs for the graphic computation of depth to a stratum will be found in a paper by Mertie .

The stratum reached is always that lying at a depth a = 1/3 C C below the current stake. This fact must always be borne in mind, since two factors, depth and the lateral position of the stake, are involved and must be considered for a proper interpretation of the data.

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6 - Mertie, J. B., jr., Graphic and Mechanical Computation of Thickness of Strata and Distance to a Stratum: U. S. Geol. Survey, Prof. Paper 129-C, 1922 pp. 39-52.

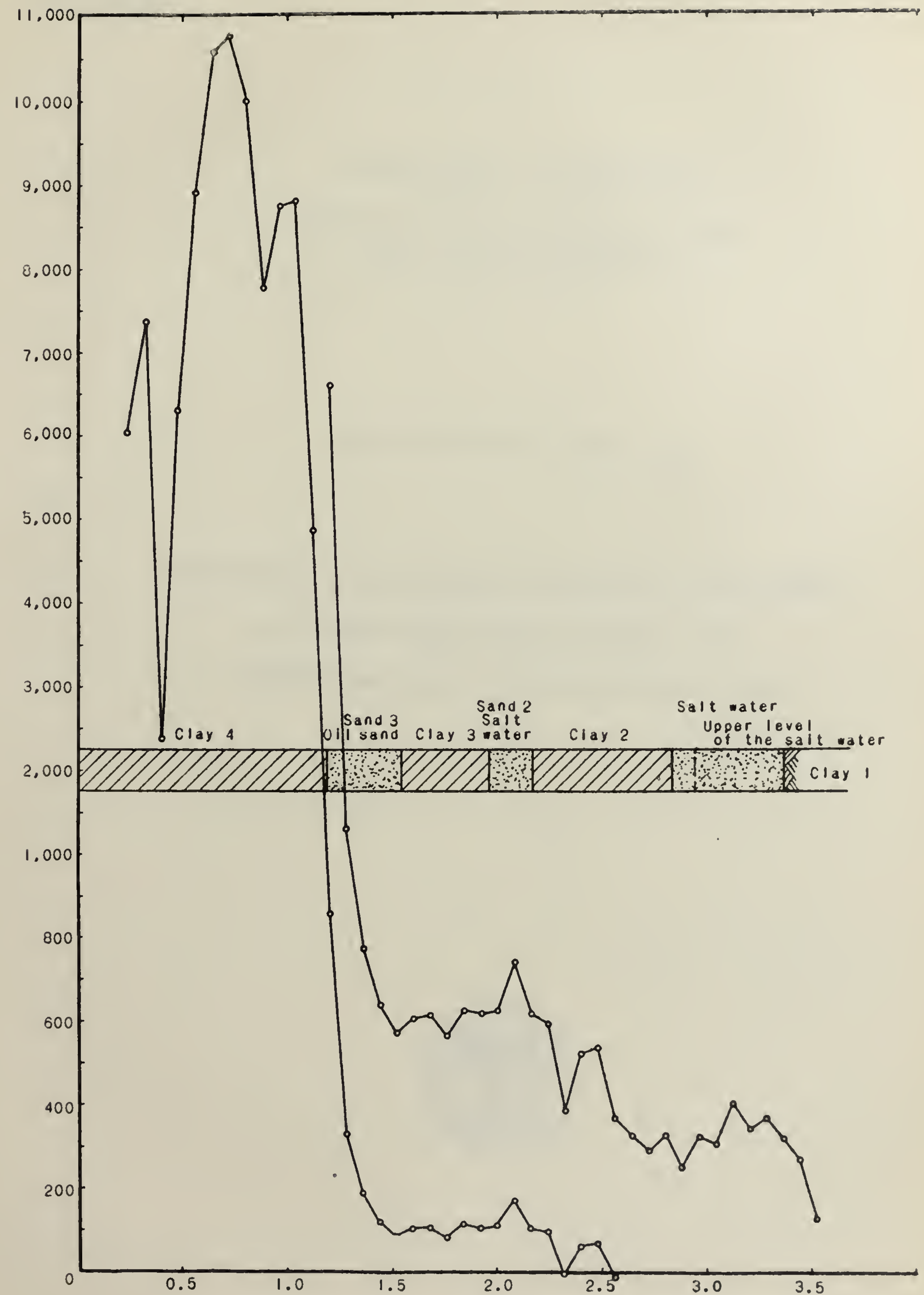

Figure 14.- Curves obtained on hilly side of the experimental blocks by the single-electrode probe, keeping a = 2 b

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MINING AND CRUSHING METHODS AND COSTS AT THE WEST PENN CEMENT CO., LIMESTONE MINE, WEST WINFIELD, PA.

BY

George A. Morrison

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE   BUREAU OF MINES

MINING AND CRUSHING METHODS AND COSTS AT THE LIMESTONE MINE OF THE WEST PENN CEMENT CO., WEST WINFIELD, PA. [1]

By George A. Morrison [2]

INTRODUCTION

This paper describing the methods of mining and crushing limestone at the West Winfield, Pa., limestone mine of the West Penn Cement Co., is one of a series being prepared by the Bureau of Mines on mining practices, methods, and costs in the various nonmetallic mining districts.

HISTORY

The limestone deposits of western Pennsylvania have been exploited for many years and have been quarried extensively at West Winfield in Winfield Township, which lies in the southeastern corner of Butler County.

At first the limestone was burned in open kilns but later when the demand for furnace stone to flux the iron ores brought into the Pittsburgh district became heavy, the larger sizes of stone were shipped to meet this demand and the smaller sizes were screened out to supply requirements for agricultural lime and railroad ballast.

As the quarries penetrated farther into the hillsides, the removal of overburden became more and more of a problem, until the cost of its removal made the economic quarrying of the stone prohibitive. Underground methods were then tried. This was about 1892, and mining has been going on continuously ever since.

The early methods of mining were naturally crude, the workings were irregular, and hand labor predominated. Later piston drills operated by gas were used until an explosion occurred which was caused by ignition of the gas. The next step was the utilization of steam for the drills and this was

1. The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgement is used: "Reprinted from U. S. Bureau of Mines Information Circular 6446."
2. One of the consulting engineers, U. S. Bureau of Mines.

followed by gas-driven air compressors. From this time on progress in mining practice was steady, and present-day operations are along modern lines.

The West Penn Cement Co. was formed about four years ago to manufacture cement from deposits at West Winfield.

TOPOGRAPHY AND GEOLOGY

The whole property lies in rough rolling hills traversed by streams flowing in a southeasterly direction to the Allegheny River. These streams have cut through the strata and exposed successive horizons of economic value. An important uplift with a northeasterly axis crosses the property. Glacial deposits are scattered over the surface of the hills.

A broad brecciated area, averaging some 1,500 feet across and showing a considerable amount of movement and shifting of the beds, crosses the property in a northeasterly direction. This area contains no workable measures and divides the property into a northern and a southern part.

The accompanying sketch (Fig. 1) shows the sequence of the strata. All mining operations are in the Carboniferous sedimentaries. The Vanport limestone and the underlying shale and sandstone are being mined while the coal that is uncovered by the removal of the shale is left for extraction at a future date.

The limestone stratum, known as the Vanport, is extensively developed throughout western Pennsylvania. It lies about 65 feet below the Lower Kittanning coal measures and is recognized by its fossils (mostly crinoids) and certain physical characteristics as being of marine origin.

On the West Penn property this limestone lies nearly horizontal but in reality it forms the top of an anticline with a north and south axis. The limestone averages nearly 25 feet in thickness on this property and is quite uniform from top to bottom. The only exception to its uniformity is a layer or two near the top which is called the "shell" layer. This layer is denser and more brittle than the underlying stone and often has a layer of shale up to an inch or more in thickness interbedded with it. The bedding planes are well pronounced throughout the deposit and certain layers part well upon blasting. Joint planes and fracture planes run nearly vertical and occasionally break up the stone on shooting, but usually the blasted material is in great masses.

The following analysis is typical of this limestone:

	Per cent
Calcium carbonate..............	90.00
Magnesium carbonate...........	1.00
Silica.........................	5.00
Alumina........................	2.50
Ferric oxide...................	1.50

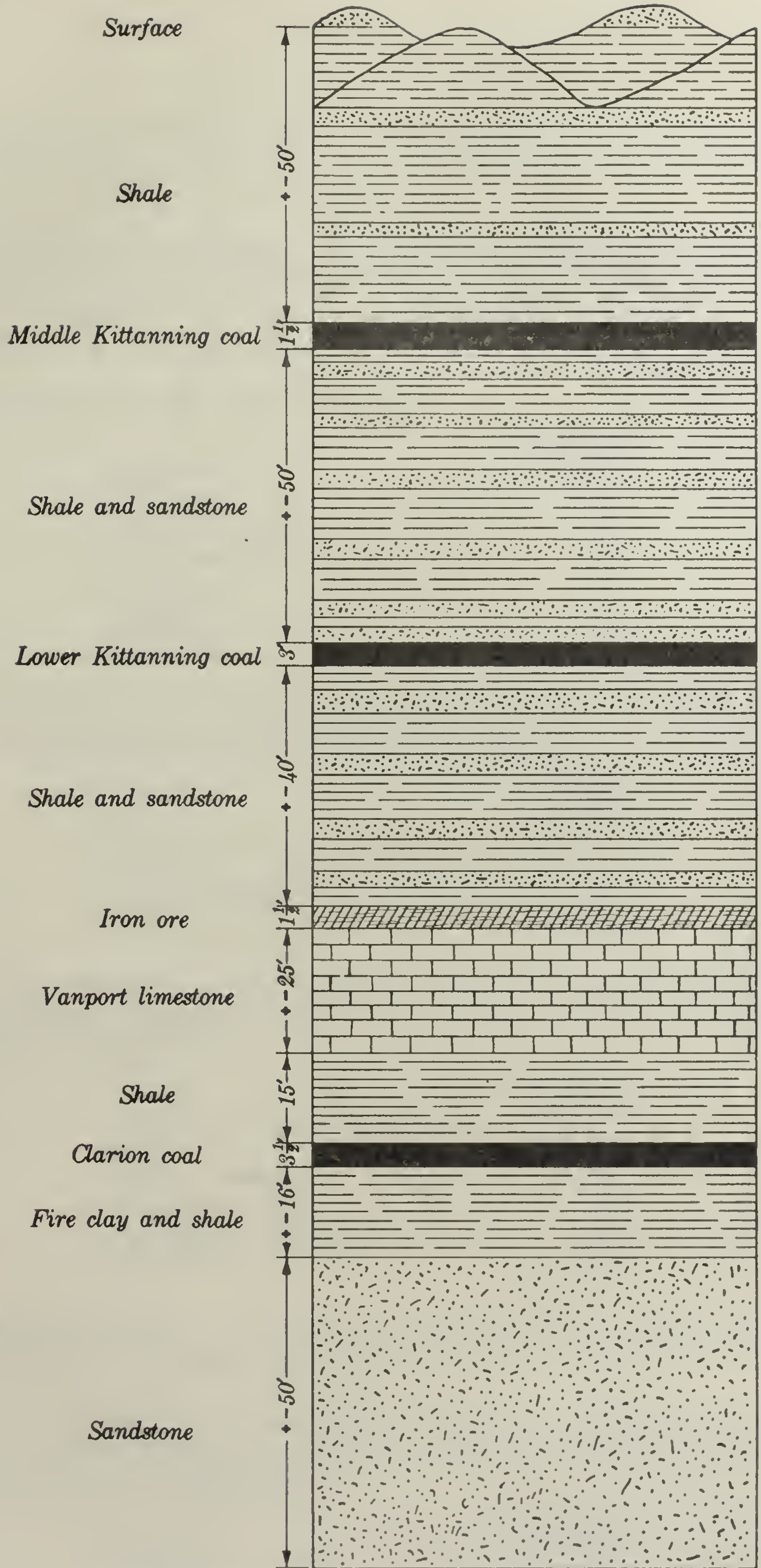


Figure 1.— Vertical section showing the relation of rocks at West Winfield, Pa.

The limestone rests upon 14 feet of shale from which it parts readily when blasted. This shale in turn rests on 3 1/2 feet of coal (Clarion) which likewise parts readily from the shale.

The shale, which is also mined, is dark gray, of uniform character, and so dense that it must be drilled and blasted for its removal. There are no fossils in this shale, but as it lies between Carboniferous strata -- the limestone above and the coal below -- it is of that age.

A typical analysis of the shale follows:

	Per cent
Silica	56.0
Alumina	19.0
Ferric oxide	8.0
Calcium oxide	.5
Sulphur	3.5
Water	6.8
Volatile	4.0

The Clarion coal which is uncovered by the mining of the shale is a fair grade of soft coal and has the following analysis:

	Per cent
Volatile	33 to 44
Ash	13 to 6
Sulphur	2 to 1
Fixed carbon	52
B.t.u.	12,600

Lying directly below this coal are from 10 to 16 feet of fire clay (known as the Clarion). This is used by the Pennsylvania Clay Products Co. and has the following analysis:

	Per cent
Silica	57.22
Alumina	34.17
Ferric oxide	.24
Calcium oxide	1.17
Magnesium oxide	.49
Water	5.82

While probably too high in calcium and magnesium for making fire brick, this clay is desirable for tile, sewer pipe, etc.

Immediately below the fire clay is the Clarion sandstone. This deposit is 50 feet thick, very uniform in character, and has a high silica content. The following is an average analysis:

	Per cent
Silica	98.28
Alumina	.35
Ferric oxide	.61

The only impurities are occasional iron spots and knife-edge seams of interbedded coal.

Overlying the limestone are repeated layers of shale and sandstone, which are of no economic value. Among these impure and weak measures are two coal beds. The lowest of these, the Lower Kittanning, is about 3 feet thick and may be of value some day.

EXPLORATION

The extensive quarrying of the limestone in the early days proved the uniformity of the Vanport limestone in the western Pennsylvania district. The numerous well drill holes driven on this property quite completely outlined the deposit on the company's land. The drill holes were primarily for gas and oil which is found at a depth of some 2,000 feet. Many of these wells are producing at present and must be avoided in mining.

Recently holes have been drilled to determine the extent of the brecciated zone already mentioned. These holes, 5 1/2 inches in diameter, are from 100 to 300 feet in depth and cost 75 cents to $2 per foot, depending upon accessibility of their location, depth, etc.

In some places the limestone outcrops, but for the most part it is covered with a thick mantle of shale, sandstone, and soil.

The uniformity of the stone as to its chemical analysis, physical characteristics, and extent simplifies the mining of the deposit.

SAMPLING

Few mine samples are necessary but samples are taken hourly and analyzed at the cement plant laboratory and careful check is kept on all ingredients as they are ground into slurry. Slight variations are corrected to maintain the desired analysis of the mix going into the cement.

A considerable amount of the stone mined is sold for commercial purposes. This is marketed on a basis of screen analysis specified by the purchaser. These analyses are made at the mine so the product may be kept up to the specified requirements.

CHOICE OF MINING METHODS

The flat-lying strata penetrate from the valley walls into steeply rising hills until the limestone measures often have an overburden of 200 or 300 feet. In the early quarrying the outcropping stone was first worked out. Afterwards stripping was carried on until its cost prohibited further operations by this method.

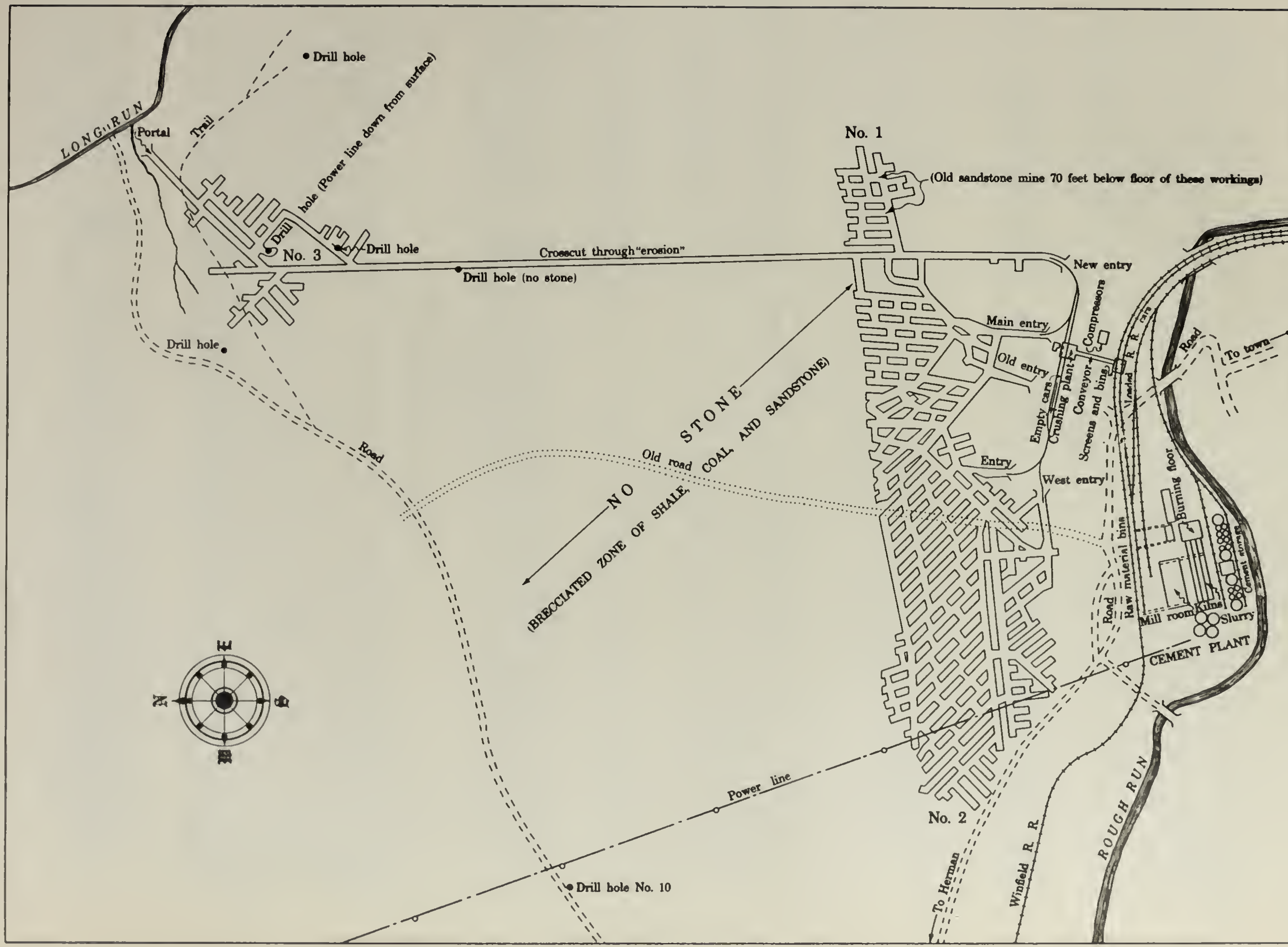


Figure 2. – Plan of mine workings

Then openings were driven into the hills for removing the stone without stripping. Some irregular pillars were left to support the roof. Soon afterwards a single-entry system was adopted, with rooms running off at regular intervals to either side. Later came the installation of the present double-entry system with rooms driven to the left off the left-hand entry and to the right off the right-hand. This system of rooms and entries with ribs left between them for support is the present practice.

The entries and rooms are 30 feet wide, and 25-foot pillars are left between. Openings 30 feet wide are put through the rib pillars every 100 feet or so.

Two or three layers of stone are left in place in the roof of these rooms and entries to support the weak overburden of shale and sandstone. The thickness of roof layers left for support varies from 3 to 5 feet, depending upon the thickness of the overburden at that point.

The floor of the workings is the top of the shale measures. The height of the workings is determined by a uniform parting layer usually occurring 19 feet above the floor of the limestone. This height is quite constant but in a few places in the mine it drops to 10 or 12 feet. The high, wide openings of the underground workings stand up well with little or no timbering.

The accompanying map (Fig. 2) shows a plan of the mine workings in relation to the surface layout.

During the busy season a daily production of 2,000 tons of limestone and about 200 tons of shale is maintained. There is no waste, as all of the tonnage broken is either screened and sold to the commercial stone trade or shipped to the cement plant.

DRILLING AND BLASTING

The method of extracting the stone is a breast-stoping system whereby the opening is advanced its full height in one cycle of operations. The width of 30 feet is taken out in a series of "V" cuts driven into the face. The center cut takes out a wedge-shaped piece to a depth of about 8 feet. Then by setting up the drill in the notch formed by the shooting of the "V" cut, holes are drilled to meet other sets of holes drilled from the original face along the rib. These are called the "angle cuts" and two of them are required to bring the full room width of 30 feet for an advance of about 8 feet.

The drill is again set up but this time on the muck pile resulting from the previous blast, and the cycle is repeated again and again, each time from a higher muck pile, until the last set-up is 12 feet or more above the floor.

The drilling is arranged so that a set of four holes fanned from one set-up will meet four holes from another set-up at right angles. The drill runner lays out the holes by using a powder box as a square and a drill-steel for a straight edge so that he can determine where to start the companion holes, their angle to the face, and their depth.

The vertical arrangement of holes is also fan shaped and drilled so four holes from one set-up will meet four holes drilled from another set-up. The drilling is done so the top holes come together about a foot below the desired roof and the bottom holes about the same distance above the shale floor. The vertical distance between the apexes of the top holes and those of the bottom holes is 15 feet, and a break 17 feet high results when the holes are blasted.

The accompanying sketches (Fig. 3) show the drill round in plan and section.

After three or four rounds the face has been advanced about 30 feet and a stoper drill is set up on top of the broken stone pile to drill 2-foot vertical holes (Fig. 3) into the roof on 6-foot centers. These holes are blasted in units of 30 to 36 holes, and the room, then containing about 2,000 tons of broken stone, is ready for the power shovel.

In drifting, 3 1/2-inch hammer drills are used mounted on tripods, and all holes are drilled wet. Air at 100 pounds pressure at the compressors delivers about 80 pounds at the drills. Line oilers and a good grade of liquid grease are used on all machines.

The drill steel is 1 1/4-inch hollow round and is fitted with lugs on shanks. The starter has a 2 1/8-inch cross bit of the McLellan type and 1/8-inch decrease in the gage is made on each 30-inch change in the steel length so the bottoming steel (13 ft. 6 in.) has a 1 5/8-inch bit.

A machine will average 165 feet of holes per drill shift making four setups and using from 6 to 9 bits. The drilling speed of the machines is 9 inches per minute, and the steel is actually cutting rock 37 per cent of the time. The balance of the drillers' time is consumed in rigging in and out, changing set-ups and steel, blowing out finished holes, and mucking out for set-ups. A helper is used on each machine, as the long steel must be bent to get it into the hole in the narrow "V" cut.

The depth of holes varies from 8 to 13 feet, depending upon the condition of the face. All drill holes are loaded by blasters on a night shift and are fired by blasting machines.

A special pulverant dynamite of 35 per cent strength is used in 1 1/2 by 8 inch cartridges for all face shots. Another type of dynamite of 37 per cent strength and less mealy is made up in 1 1/4 by 8 inch cartridges for use in blockholes and stoper work. Little secondary blasting is required,

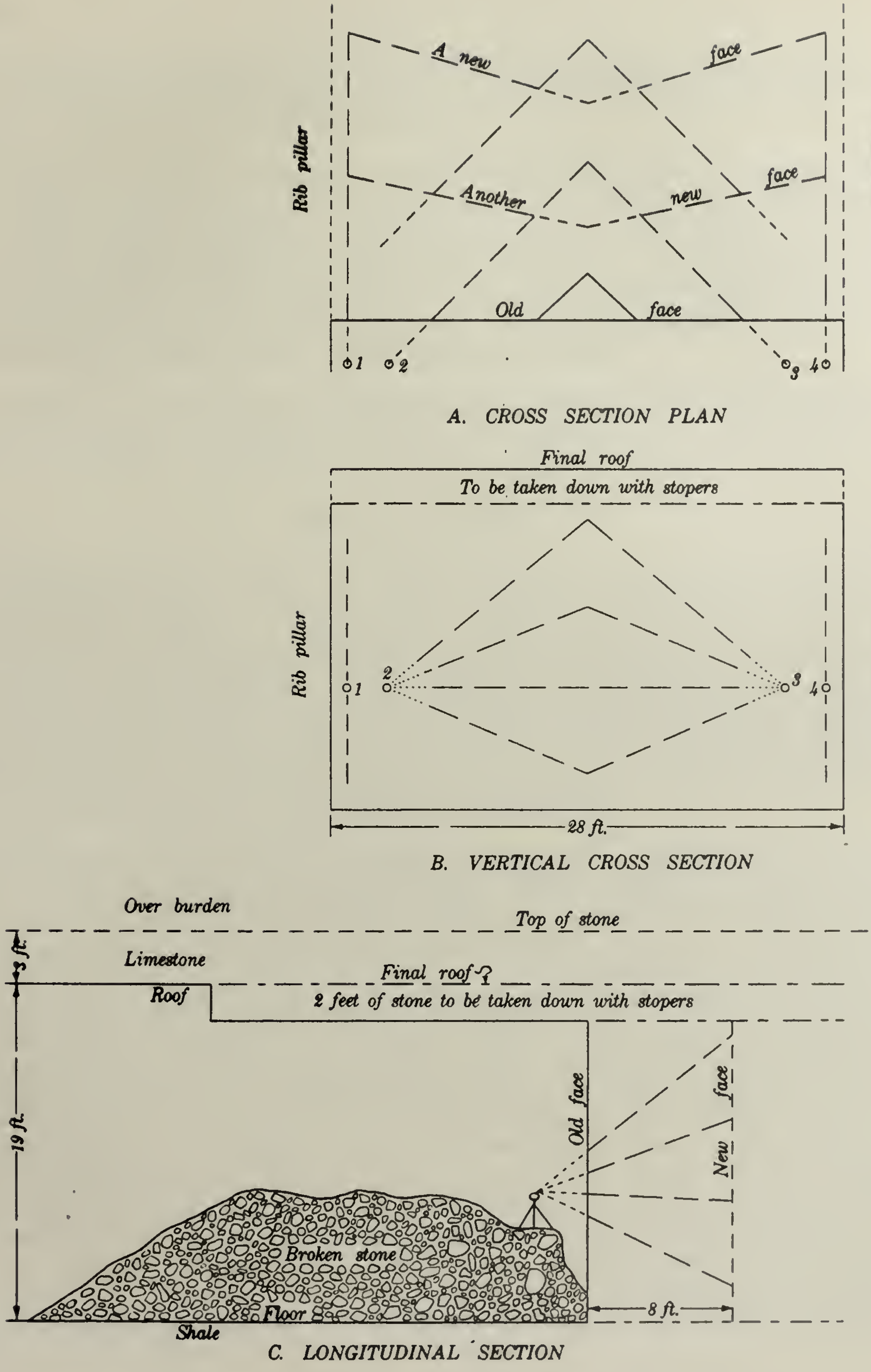


Figure 3. – Drill round

as many face holes are intentionally overloaded so the blasted material is kicked as far back into the room as possible to insure its being filled with broken stone before the shovel is moved in.

All holes are tamped with at least three cartridges of tamping made up by the blasters. No. 6 electric detonators with 8 and 10 foot copper lead wires are used. Three blasters working on a night shift prepare the tamping, make up their primers, and load and shoot all holes. No. 14 insulated copper wire is carried on fixed supports to all faces being worked from locations convenient for the blasting stations. No. 20 annunciating wire is used to connect the main lead wires to the detonator wires.

An average of 0.6 pound of dynamite is required per ton of stone produced. This includes all secondary shooting and powder used for any other purpose.

LOADING THE STONE

The broken stone is loaded by means of three full-revolving electric shovels which are mounted on caterpillars. These shovels are of 5/8, 1, and 1 1/4 cubic yards capacity, respectively. The 5/8 cubic yard shovel is several years old and inefficient but the other two shovels are of modern design and have proved very efficient. Each of these newer machines is fitted with a 15-foot boom and a 10-foot dipper stick. The first two shovels use 250-volt current supplied from the trolley lines, and the last and largest shovel uses alternating current at 440 volts which is converted to 250 volts (d.c.) by means of a motor generator set mounted on the shovel.

Twenty three hundred volt alternating current is brought into the mine through a 12-inch drill hole 140 feet deep. This current is stepped down at the bottom of the hole to 440 volts before being carried in well-insulated triple-conductor cable through the mine to suitable taps for connecting flexible feed wires to the shovel. The largest shovel has proved especially satisfactory and has repeatedly loaded over 1,000 tons of stone in a 10-hour shift. In 1930 it averaged 700 tons per shift. An average for the three shovels combined is 1,500 tons per shift. The shovels clean up well along the ribs and at the faces. Each is served by a trolley locomotive equipped with a gathering reel and running on double track in each entry and room. These locomotives keep an empty car at the shovel on one track while replacing a loaded car with an empty one on the other track. Enough empty cars are kept in the string at the shovels so, as is often the case, more than one car can be loaded before requiring replacement with empties. When a train of 7 to 12 cars has been loaded, it is hauled away to the crusher and an empty train is moved up to the shovel. In the meantime the shovel has from two to four empty cars to load which were left by the locomotive before making its trip.

Table 6 shows the loading data for the first six months of 1930.

Hand loaders are employed at a contract price of \$0.215 per ton to clean up faces and clean out rooms where track is to be laid up to the blasted pile so the power shovels can work on a thick pile of broken stone. At present the hand-loaded tonnage amounts to about 17 per cent of the total tonnage. Later another power shovel will be put to work and much of this hand labor will be eliminated.

A typical shovel crew consists of the following men:

Drillers	2 - 4
Drillers' helpers........	2 - 4
Scalers and blasters.....	2 - 3
Locomotive runners.......	1 - 2
Locomotive-runners' helpers	1 - 2
Trackmen................	2 - 4
Shovel runners..........	1 - 1
Shovel-runners' helpers..	1 - 1
Total...............	12 - 21

TRANSPORTATION

Seven-ton trolley locomotives are used, and at present there are four on the property. One locomotive is used for each shovel, and the fourth is used to haul to the crusher from a remote part of the mine. At present the locomotives gathering for two of the shovels also haul to the crusher.

These locomotives have 22-inch wheels with a 4-inch face, a 44-inch wheel base and are 13 feet long over-all. They are powered with two 120-ampere, 250-volt motors, are driven by worm-gears, and have a draw-bar pull of 3,700 pounds. Previously horses were used for gathering but in 1930 their use was entirely discontinued.

The mine cars (Fig. 4) are a solid-body type dumped by means of a rotary car-dumper, which is actuated by compressed air. These 135 cubic feet capacity cars have 4-inch axles, with outside journal boxes mounted on 8-inch Hyatt roller bearings and are equipped with four-wheel brakes. The car is 12 feet long by 6 feet wide and stands 46 inches above the track. The wheels are 16-inch chilled iron having a face of 4 inches and a wheel base of 38 inches. One wheel is tight on each axle. The car bottom is of 3-inch oak plank between 3/8-inch sheet-iron plates. The cars weigh nearly 4 tons each and carry an average load of 8 tons. The couplings are swivel-hitchings of heavy design, so a train can be dumped at the crusher without uncoupling the cars.

Trains of from 8 to 12 cars traverse a circuit so arranged that they are always moving in one direction.

Tracks of 36-inch gage are laid with 40-pound rails on 5 by 6 inch by 5 foot oak ties on 2 1/2 foot centers. Every fourth tie is a steel one of heavy design having fixed gage clamps. Curves are of 70-foot radius. No. 3 frogs are used in the switches with 5-foot switch points and 7 to 9 foot

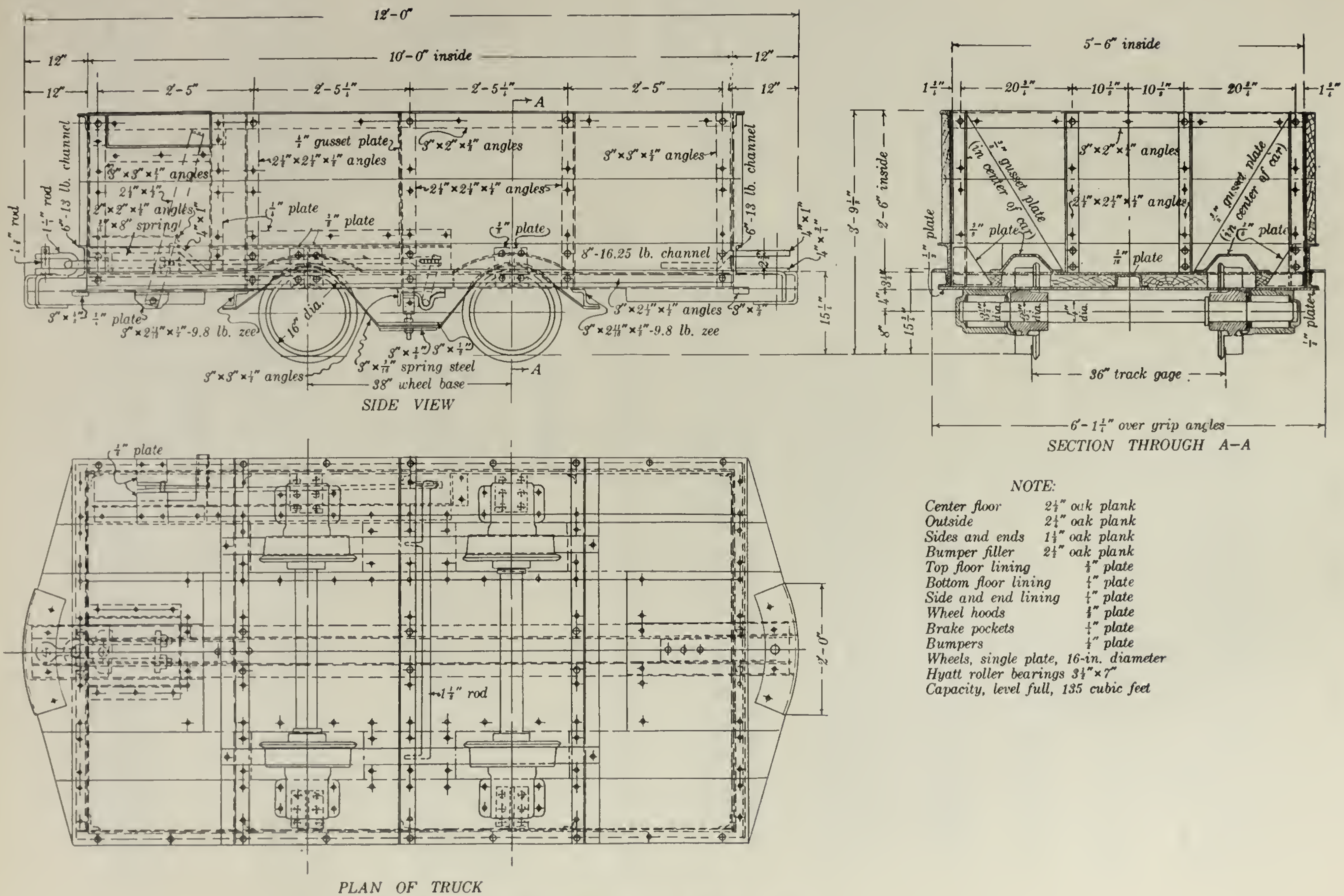


Figure 4.—Mine car of the West Penn Cement Co.

lead rails. The floors are usually smooth and level but occasional rolls occur.

Service tracks to the working faces are of a temporary nature and are made up of 10, 15, 20, and 30 foot sections mounted on light steel ties so they can be removed and carried to another working face or else relaid when required after blasting. Rock ballast is used on all permanent track, and every effort is made to maintain all such track in the best condition.

Parallel throw-switch stands are used to avoid the possibility of the operator being thrown under passing cars when throwing a switch.

In 1930 the average haul (loaded) was about 3/4 of a mile and the 75 cars serving the shovels averaged five trips per shift.

Car repairs at present are mostly due to broken wheels. A new wheel has been designed and its use is expected to reduce this breakage greatly.

The trolley is a "0000" wire suspended 8 1/2 feet above the track by means of "low-mine" suspension insulators and pipe trusses fastened into the roof by expansion sleeves. By careful bonding and crossbonding a current of 250 volts is maintained throughout the mine.

All power is purchased, at 2,300 volts and transformed to 250 volts direct current for haulage in the mine by means of a 150-kilowatt motor-generator set of modern design which is in turn served by an automatic switchboard.

Ten to 12 men are engaged in serving the shovels and maintaining track.

DRAINAGE

Little water is encountered, but in certain sections of the mine there is some water at the faces and it is a great hindrance in laying track and in screening the wet stone in the crushing plant. Small pumps operated by air are installed where necessary and moved from place to place as required.

VENTILATION

All ventilation is natural and is obtained by frequent openings to the surface. These openings are of a large cross section. The constant clearing away of all rubbish and the confining of blasting to one shift contribute to maintaining excellent working conditions.

TIMBERING

Occasional timbering is required to support certain layers in the roof that may become loose. Ten to 12 inch posts cut from the company's property are used with heavy headboards. At some places in the mine the openings to

the surface pierce roof layers so badly leached that they have no strength. In such places drift sets with heavy lagging are required.

SHALE MINING

The floor of the limestone is shale which extends from about 14 feet to a 42-inch bed of coal. This shale is mined out of rooms from which the stone has been extracted.

To reach this level at the top of the coal a track on a 7 degree slope is carried down from the old room floor at one place in the mine. The track is thereafter continued on the top of the coal from room to room, going through ribs as they are encountered at points where holes have previously been cut in mining the limestone.

The shale is broken with sinker drills which drill 12-foot holes on 6-foot centers in the old room floors. About 250 tons of shale are required per day and this is broken by two men who do all drilling and blasting and look after the track. The shale loading is all done by hand. The loaders are paid on contract at a rate of $0.215 per ton.

The coal uncovered by mining the shale is left in place for mining at some future date when it may be required. A considerable tonnage of this coal has already been uncovered.

CRUSHING PLANT

Car-Dumper.- The loaded cars, delivered by locomotives in trains of from 8 to 12, are set on the loaded-car track at the crushing plant. An electrically driven car-puller, consisting of a standard hoist handling a continuous cable through sheaves along this track, hauls the trains in over the car scales and into the car dumper, which is set on a 1 1/2 per cent slope. The cars are completely emptied without being uncoupled by being given a half turn in the dumper cradle, which is revolved by means of compressed air. Two men serve the dumper and also hook the trains and look after the crushers below. The empty cars drift down the empty-car track to be picked up by the locomotive that brought in the loads and which has by-passed the crusher building. The return trip into the mine is on a different track than that over which the loads came out and a circular path of traffic is thereby maintained over which the trains are moving only in one direction.

The dumped stone goes directly into a 30-inch gyratory crusher with its discharge opening set at 4 inches and which is belt driven by a 125-hp. motor. The product from this primary crusher goes over a No. 9 live-roll grizzly with 2 3/4-inch openings which is powered by a 10-hp. motor driving through sprockets and chain.

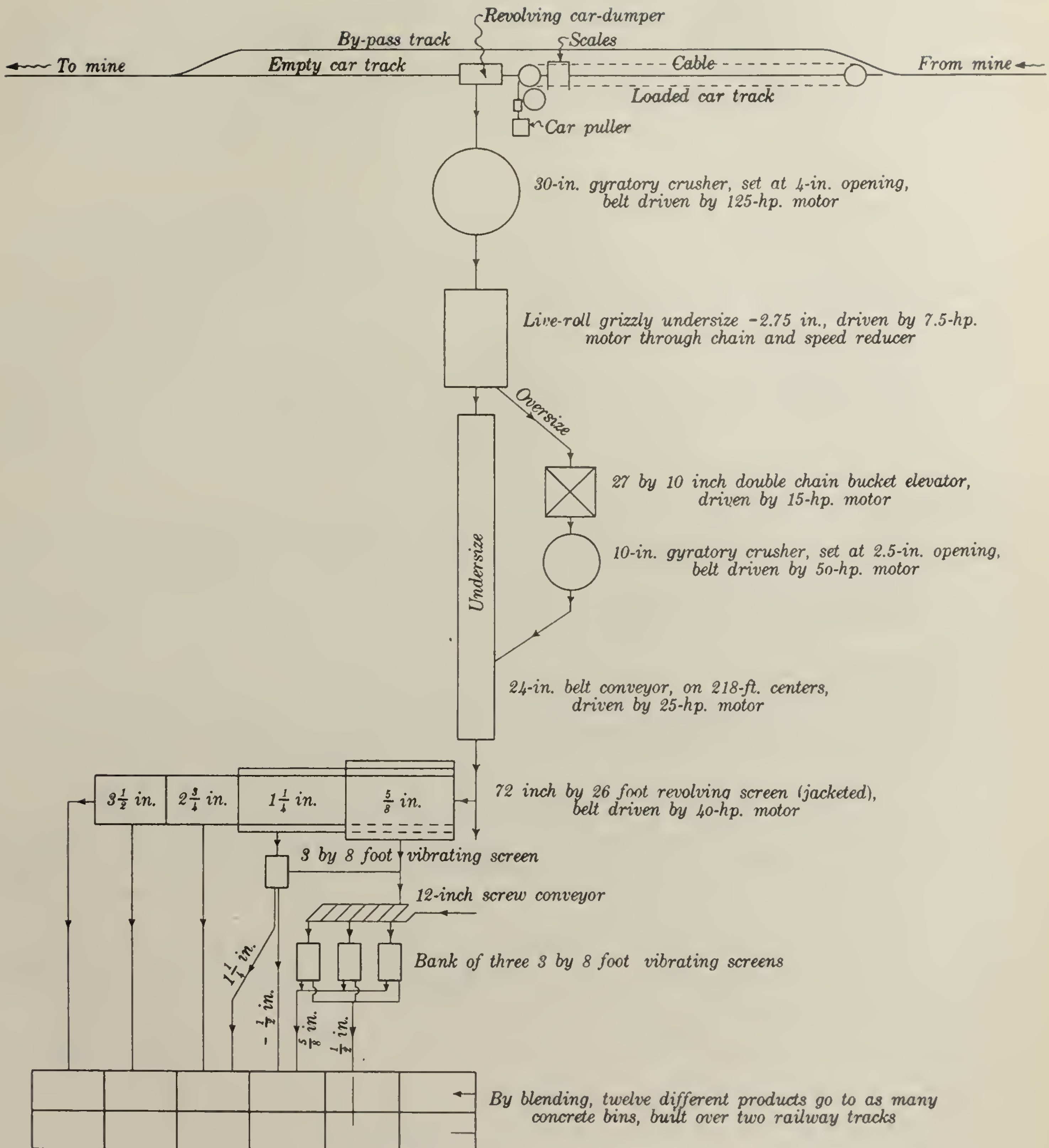


Figure 5.—Flow sheet of crushing plant

From the grizzly the oversize is lifted by means of a 40-foot elevator equipped with close-connected steel buckets on double chains, to a 10-inch gyratory crusher which is set at a 2-inch discharge opening and is belt-driven by a 40-hp. motor. The discharge from the crusher joins the undersize from the grizzly and is delivered onto a 24-inch conveyor belt 218 feet long and inclined 3 1/8 inches per foot which conveys the stone to the top of the screen room.

In the screen room is a double-jacketed revolving screen 72 inches in diameter and 26 feet long, mounted at a slope of 1 1/4 inches to the foot, over concrete bins. The holes in the revolving screen are round and are punched in 3/8 inch plate. The first jacket is also made of punched plate but the second one is of woven wire with 5/8-inch holes. All vibrator screens are also of wire screening with square holes. The products of the revolving screen, after the proper blending on vibrating screens, fall into 12 concrete bins which hold about 150 tons each. The bins feed through wide-mouthed gates, operated by hand, into railway cars standing on the double railway tracks immediately beneath.

The delivery of the product carried by the conveyor from the crushers is arranged so that, by means of chutes and gates, stone for the cement plant can be delivered directly to bins without having to go through the revolving screen. The screened products are blended as required by the specifications of the trade for road material and concrete aggregate.

An 18-inch apron conveyor traverses one side of the bins and elevates material from any bin to a mechanically-vibrated screen which is drenched with water from numerous nozzles under high pressure. This washed material is drained while passing through metal spouts into the railway cars below.

Finished stone for the cement plant is delivered in hopper-bottomed railroad cars which are drawn into the unloading-tunnels at the plant, or else have their loads stocked in convenient piles to be reclaimed by locomotive cranes as required.

Truck haulage from all bins is provided by conveniently operated chutes. Figure 5 is a flow sheet of the plant.

Table 1 gives a summary of crushing data for the first eight months of 1930.

Table 1. - Crushing Plant Operating Data, First Eight Months of 1930.

Month	Tons	Total hours	Delays in hours due to				
			No stone	No railroad cars	Big pieces	Miscel-laneous	Total
January	33,800	195	16	1	3.5	3	23.5
February	26,400	144	15	.5	1	4	20.5
March	37,000	200	24	1.5	3	6	34.5
April	23,650	152	32.5	--	3.5	.5	36.5
May	42,500	250	30.5	--	5	2	37.5
June	51,700	270	25	--	3	3	31
July	48,200	231	32	1	4	5	42
August	51,975	262	40	--	6.5	1.5	48
Total	315,225	1,704	215	4	29.5	25	273.5
Percentage of delays	--	--	78.5	1.5	11	9	100

Percentage of lost time - 16
Average number of tons per hour - 186
Average number of tons per hour (if no lost time) - - - - - - - - - - - - 220

Note:

"No stone" means that the dumper was idle because of no loaded mine cars to dump.

"No railroad cars" means that there was a shortage of these cars.

"Big pieces" indicates that the crusher was plugged or arched over.

Under "Miscellaneous" are included delays for oiling, repairs, or cars with broken wheels.

The tonnage is for stone and shale. The shale is much slower in crushing.

The plant capacity can be assumed to be 250 tons per hour, and the actual production 225 tons.

In the three years this plant has operated, no renewals have been required by the crushers (except for some bottom plates in the 30-inch gyratory), conveyor, or revolving screen. Chutes have had to be relined and patched, however. The screens on the vibrators last about six months, and the drive belts on them are likely to whip out at the fastenings in a few months.

The flow sheet (Fig. 5) shows that the undersize through the 5/8 inch revolving screen passes to a bank of three vibrating screens. A smaller number of vibrators might do the work but it was considered possible that 1/8-inch screens might be used on them, and as the stone from the mine is often damp it is necessary to carry a thin bed on these machines to insure proper sizing.

Table 2 shows screen analyses of the various products.

The crushing plant is operated lrimarily to supply the cement plant with raw material and the screening plant to supply surplus production to the commercial stone trade. The requirements as to size of commercial stone vary and the tonnage demand is also erratic. No anticipation of the sizes and tonnage required can be made definitely. Since space for stock piling is limited around the plant and the demand for the products fluctuates, it is often necessary to put the whole daily production over the screens to produce the required tonnage of some certain size or sizes.

Table 3 lists some of the products sold and their specifications. By means of spouting between screens and bins, mixtures are made so many combinations can be turned out for shipment.

PER CENT EXTRACTION

As there is no concentration, the crushing plant delivers the same tonnage it receives.

The mining recovery at present is 55 per cent. The remaining 45 per cent can be recovered later by removing pillars on a retreating system from any places where the surface is of so little value that subsidence is not objectionable.

The cement plant consumes two-thirds of the mine output and the balance is sold to the trade as commercial stone.

EMPLOYEE'S PAY SYSTEM

The mine is operated on a six-day week basis of 8 or 10 hours daily, depending upon the tonnage required. Due to a fluctuating market, the working season varies somewhat, but 10 months is about the average year.

Except for about 18 per cent of the total, all labor is paid on an hourly basis, and wages vary from 45 to 80 cents per hour. The shale and some of the stone is loaded by hand on contract at a rate of $0.215 per ton, and it is this labor that accounts for the 18 per cent exception noted above.

SAFETY METHODS, ETC.

Well-equipped first-aid cabinets are kept at various places throughout the mine and crushing plant. Each locomotive and shovel also carries a kit.

The mine is divided into three divisions, each under a separate boss. These divisions compete with each other for quantity of production and the least number of accidents.

Three first-aid teams are kept in training, one team for each division of the mine.

In 1929 the whole force was given a course in first-aid under the direction of a representative of the Bureau of Mines.

Careful scaling of the roofs, well-planned preparation for shots, adequate lighting of workings, cleaning up openings and keeping passageways free from stone, etc., maintaining equipment in the best repair, and efficient workmen who think of the possibilities of an accident as well as efficiencies all aid in maintaining the low accident rate the company enjoys.

EFFICIENCIES AND COSTS

Tables 4, 5, 6, 7, and 8 give various statistics of performance. Figure 6 is an organization chart.

Table 2. - Screen Analyses

	Primary crusher				Secondary crusher				Main screen		5/8-inch screen	
	Mine run (feed)		Product		Feed		Product		Feed		Feed	
Percentage of total	100		100		35.75		35.75		100		17	
Tons per hour	225		225		80.50		80.50		225		38.25	
	Per cent	Tons	Per cent	Tons	Per cent	Tons	Per cent	Tons	Per cent	Tons	Per cent	Tons
Plus 6-inch	25	56.25	0	--	--	--	--	--	--	--	--	--
Minus 6 plus 3½-inch	15	33.75	[1]6	13.50	[1]71.5	57.56	[1]5.8	4.67	2.0	4.50	--	--
Minus 3½ plus 2¾-inch	19	42.75	32	72.0	17.5	14.09	[1]33.0	26.57	12.0	27.0	--	--
Minus 2¾ plus 2-inch	12	27.00	14.8	33.3	8.5	6.84	32.1	25.84	25.0	56.25	--	--
Minus 2 plus 1¼-inch	10	22.50	18.0	40.50	1.5	1.21	21.5	17.30	26.0	58.50	--	--
Minus 1¼ plus 5/8-inch	10	22.50	[2]15.0	33.75	.5	.40	3.8	3.06	18.0	40.50	--	--
Minus 5/8 plus 3/8-inch	2	4.50	3.0	6.75	--	--	1.0	.80	4.0	9.00	21.2	8.10
Minus 3/8 plus 1/8-inch	3	6.75	[2]5.6	12.60	.25	.20	1.4	1.13	6.5	14.625	38.8	14.85
Minus 1/8-inch	4	9.00	[2]5.6	12.60	.25	.20	1.4	1.13	6.5	14.625	[2]40.0	15.30
	100	225	100	225	100	80.5	100	80.5	100	225	100	38.25

1 Mostly flat tabular pieces.
2 Increase in percentage due mostly to spalling and wearing down in chutes.

Table 3. - Screen analyses of various crushing-plant products.

Mater-ial	Percentage through												
	100 mesh	1/8 inch	1/4 inch	3/8 inch	5/8 inch	1 inch	1-1/4 inch	1-1/2 inch	2 inch	2-1/2 inch	2-3/4 inch	3-1/2 inch	4 inch
A	10 - 25	-	-	-	100	-	-	-	-	-	-	-	-
B	-	0 - 10	0 - 20	-	100	-	-	-	-	-	-	-	-
C	-	-	-	0 - 8	10 - 40	-	95 - 100	100	-	-	-	-	-
D	-	-	-	0 - 10	0 - 50	95 - 100	100	-	-	-	-	-	-
E	-	-	-	0 - 8	5 - 20	-	30 - 65	-	-	95 - 100	100	-	-
F	-	-	-	-	-	-	0 - 15	-	30 - 65	95 - 100	100	-	-
G	-	-	-	-	-	-	0 - 10	-	-	25 - 50	-	90 - 100	100
H	-	-	-	0 - 8	0 - 30	-	95 - 100	100	-	-	-	-	-
I	-	-	-	0 - 8	0 - 15	-	30 - 65	-	-	95 - 100	100	-	-
J	-	-	-	-	-	-	0 - 15	-	-	30 - 65	-	95 - 100	100
K	-	0 - 3	0 - 10	25 - 40	Pass $\frac{1}{2}$"	-	-	-	-	-	-	-	-
				25 - 40	" $\frac{3}{4}$"	25 - 40	90 - 100	100	-	-	-	-	-
L	-	-	-	0 - 8	0 - 15	-	30 - 65	-	-	95 - 100	100	-	-
M	8 - 12	-	-	-	95 - 100	-	-	-	-	-	-	-	-
N	-	-	0 - 5	-	0 - 25	-	-	75 - 95	100	-	-	-	-
O	-	-	-	0 - 8	0 - 15	-	30 - 65	-	95 - 100	100	-	-	-
P	-	-	95 - 100	100	-	-	-	-	-	-	-	-	-

Note:- These products are not made at the same time, but the plant may be called upon at any time to make a number of them. They are made by blending in chutes and bins, so that the proper mixture goes into the railroad cars.

Table 4. - Drilling data, January 1st to December 1st, 1929.

Periods by quarters	First	Second	Third	Fourth	Total
Drill shifts (drifters only) - - - - -	207	525	607	266	1,606
Feet of drilling (drifters only) - - -	29,874	84,823	102,000	42,526	259,220
Average feet drilled per drill shift -	145	161	168	160	160
Steel used (total number)- - - - - - -	1,315	4,584	5,609	2,420	13,928
Steel used per drill shift (average number) - - - - - - - - - - - - - - - -	6.3	8.8	9.5	9	8.7
Percent of steel broken - - - - - - -	5.3	6.0	7.5	4.7	6.3

Total shifts, all drills -	[1]2,283
Total repairs, all drills (including tanks, tripods, etc.) -	$2,107
Repairs per drill shift -	$0.92
Tons produced -	388,285
Feet drilled per ton -	0.67
Tons per drill shift (1606 plus stoper 425 equals 2031 shifts)	190
Steel-sharpener shifts -	250
Steel-sharpener shifts per drill shift - - - - - - - - - - - -	0.11
Steel used per drill shift (average number) - - - - - - - - -	8.7
Steel produced per sharpener shift - - - - - - - - - - - - - - -	55
Repairs per sharpener shift - - - - - - - - - - - - - - - - - - -	$0.60
Drill repairs per ton produced, 1929 - - - - - - - - - - - - - -	$0.0054
Drill repairs per ton produced, 1928 - - - - - - - - - - - - - -	.0053

[1]This figure includes 252 drill shifts employed in block-holing or secondary drilling.

Note: The steel in 1 1/4 and 7/8 inch sizes contained 0.87 per cent carbon. Fifty per cent of the breakage is in the 7/8-inch steel used in stoper machines.

Table 5. - Transportation data, January 1st to July 1st, 1930.

Tons hauled, stone and shale - 212,116
Tons hauled, stone and shale (including double haul in gathering - 292,100
Average length of haul, feet (loaded) -- - - - - - - - - - - - - - - 2,359
Ton-miles hauled - 130,500

Costs (gathering included)

	Per cent	Per ton produced	Per ton double-hauled	Per ton-mile
Locomotives:				
Operating labor		0.027	0.0196	0.0438
Repair labor		.004	.0038	.0062
Operating supplies		.001	.0007	.0015
Repair supplies		.003	.0025	.0055
Power		.003	.0021	.0047
Total locomotives	34	.038	.0287	.0617
Trolley:				
Labor		.004	.0032	.0071
Supplies		.002	.0016	.0035
Total trolley	5	.006	.0048	.0106
Cars:				
Labor		.005	.0038	.0085
Supplies		.006	.0041	.0092
Total cars	10	.011	.0079	.0177
Track:				
Labor		.038	.0275	.0609
Supplies		.015	.0111	.0247
Total track	48	.053	.0386	.0856
Livestock: Labor and supplies	3	.004	.0032	.0068
Total transportation	100	.112	.0832	.1824

Note: The average tons per trip to the crusher is 64.
Gathering requires many trips of but small tonnage; making up trains and placing empties. The average car load is 8 tons.

Table 6. - Power shovel data, January 1st to July 1st, 1930.

Shovel number	No. 1	No. 2	No. 3	Totals and averages
Total tons loaded	61,000.	72,500.	26,700.	160,200.
Tons loaded daily (10 hrs.)	555.	702.	342.	550.
Tons loaded per car	7.75	7.73	8.15	7.80
Percentage of time operating	84.	85.	74.	83.
Percentage of lost time	16.	15.	26.	17.
Loading cost in dollars per ton:				
Operating labor	0.028	0.028	0.047	0.037
Repair labor	.005	.006	.014	.007
Operating supplies	.002	--	--	.001
Repair supplies	.008	.007	.015	.008
Power	.004	.003	.006	.004
Total cost	.048	.044	.082	.052

Operating data:	No. 1 Per-cent	No. 1 Minutes per shift	No. 2 Per-cent	No. 2 Minutes per shift	No. 3 Per-cent	No. 3 Minutes per shift	Totals Per-cent	Totals Minutes per shift
Delays due to -								
Mechanical trouble	11)	25	20)	24	30)	59	19)	31
Electrical trouble	15)		7)		8)		10)	
Moving shovel	11	11	20	18	16	25	17	17
Track-laying, etc.	26)		20)		9)		21)	
No empties	12)		16)		20)		15)	
Locomotives	5)	60	4)	48	2)	72	4)	54
Wrecks	12)		8)		10)		10)	
Bench (hard digging)	8)		1)		5)		4)	
Total delays	100	96	100	90	100	156	100	102

	No. 1	No. 2	No. 3	Totals and averages
Maximum hourly production (tons)	75.	105.	45.	79.
Average hourly production (tons)	55.5	70.2	34.2	55.0
Efficiency: Average divided by maximum (per cent)	74½	67.	76.	70.

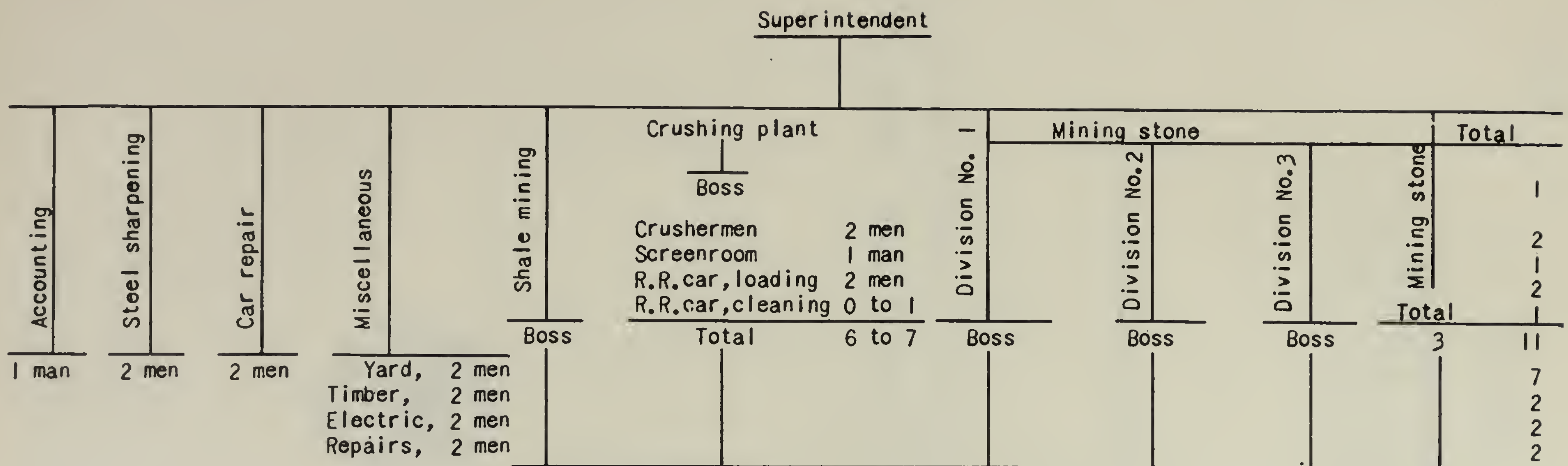
Note: All shovels are electric, full-revolving and served in the same manner. No. 1 shovel is of 1 cu. yd. Dipper capacity. No. 2 shovel is of 1¼ cu. yd. Dipper capacity. No. 3 shovel is of 5/8 cu. yd. Dipper capacity.

Table 7. - Summary of costs in units of labor, power, and supplies.

	Mining			Crushing			Other			Total		
	Stone	Shale	Stone and Shale	Stone	Shale	Stone and Shale	Stone	Shale	Stone and Shale	Stone	Shale	Stone and Shale
A.-Labor; (Man-Hrs per ton)												
Drilling	0.119	0.051	0.111	-	-	-	0.012	0.006	0.012	0.131	0.059	0.123
Blasting	0.017	0.010	0.016	-	-	-	-	-	-	0.017	0.010	0.016
Scaling	0.032	0.020	0.031	-	-	-	0.006	0.003	0.005	0.038	0.023	0.036
Loading;												
(A) Shovels	0.031	-	0.027	-	-	-	0.005	-	0.004	0.036	-	0.031
(B) Hand	0.075	0.329	0.105	-	-	-	-	-	-	0.075	0.329	0.103
Hauling;												
(A) Direct	0.049	0.050	0.050	-	-	-	0.032	0.033	0.032	0.082	0.082	0.083
(B) Track	0.062	0.036	0.058	-	-	-	-	-	-	0.062	0.036	0.058
Miscellaneous	-	-	-	-	-	-	0.025	0.025	0.025	0.025	0.025	0.025
Supt.	0.023	0.024	0.023	-	-	-	-	-	-	0.023	0.024	0.023
Crushing	-	-	-	0.032	0.044	0.033	-	-	-	0.032	0.044	0.033
Total labor	0.408	0.520	0.421	0.032	0.044	0.033	0.080	0.069	0.078	0.520	0.633	0.532
Average tons per man-shift	24.3	19.3	23.8	310.	227.	304.	125	144	129	19.2	15.8	18.7
Labor per cent of total	-	-	-	-	-	-	-	-	-	47.5	72.5	49.6
B.-Power and Supplies												
Explosives, lbs. per ton	-	-	-	-	-	-	--	-	-	.685	.466	.655
Total power, Kw. h. per ton	-	-	-	-	-	-	-	-	-	-	-	2.17
Shovels	-	-	0.41	-	-	-	-	-	-	-	-	.41
Locomotives	-	-	0.70	-	-	-	-	-	-	-	-	.70
Compressors	-	-	0.39	-	-	-	-	-	-	-	-	.39
Dump	-	-	-	-	-	0.02	-	-	-	-	-	.02
Crushers	-	-	-	-	-	0.52	-	-	-	-	-	.52
Screens	-	-	-	-	-	0.06	-	-	-	-	-	.06
Elevators	-	-	-	-	-	0.02	-	-	-	-	-	.02
Conveyors	-	-	-	-	-	0.04	-	-	-	-	-	.04
Washing	-	-	-	-	-	0.01	-	-	-	-	-	.01
Other supplies, per cent of total		-	-	-	-	-	-	-	-	-	-	10.7
Supplies and power, per cent of total		-	-	-	-	-	-	-	-	-	-	14.1
Per cent of total cost	-	-	-	-	-	-	-	-	-	-	-	78.5

Table 8. - Detailed summary of costs, January 1st to July 1st, 1930.

	Stone		Shale		Stone and shale	
	Per cent	Cost per ton	Per cent	Cost per ton	Per cent	Cost per ton
Loading:						
Total all shovels - - - - - - - -	--	$0.045	--	--	--	$0.039
Total all hand-loading - - - -	---	.045	--	$0.215	--	.075
Total loading - - - - - - - -	15.3	.090	36.3	.215	17.5	.104
Drilling:						
Operating labor - - - - - - - -	--	.098	--	.064	--	.095
Air - - - - - - - - - - - - - - - -	--	.010	--	.010	--	.010
Operating supplies - - - - - -	--	.011	--	.012	--	.012
Repair supplies - - - - - - - - -	--	.006	--	.006	--	.006
Total drilling - - - - - - - -	21.3	.125	15.5	.092	20.6	.122
Blasting:						
Labor - - - - - - - - - - - - - -	--	.013	--	.007	--	.010
Explosives and other supplies -	--	.095	--	.063	--	.092
Total blasting - - - - - - - -	18.3	.108	11.8	.070	17.2	.102
Haulage:						
Locomotives - - - - - - - - - - -	--	.043	--	.043	--	.043
Trolley - - - - - - - - - - - - -	--	.006	--	.006	--	.006
Cars - - - - - - - - - - - - - - -	--	.011	--	.011	--	.011
Track maintenance - - - - - - - -	--	.053	--	.063	--	.054
Total haulage - - - - - - - -	19.1	.113	20.9	.123	19.1	.114
General charges - - - - - - - - - -	26.1	.154	15.5	.092	25.6	.151
Grand total - - - - - - - - - - - -	100	.590	100	.592	100	.593



Shale mining		Division No. 1	Division No. 2	Division No. 3	Mining stone Total	Total
-	Stoper drillers	-	1	1	2	2
2	Drillers	2	3	5	10	12
-	Drill helpers	2	3	5	10	10
-	Blockholers	1	1	3	5	5
-	Scalers	1	2	3	6	6
-	Blasters	1	1	1	3	3
-	Shovelmen (power)	2	2	2	6	6
-	Trackmen	2	4	4	10	10
1	Locomotive runners	1	1	1	3	4
1	Locomotive brakemen	1	1	1	3	4
10	Loaders (hand)	3	3	4	10	20
-	Miscellaneous	-	1	1	2	2
15	Total mining	17	24	32	73	88
	Total outside (additional)	(Mine bosses included in total mining)				20
	Grand total					108

Figure 6.- Organization chart

I. C. 6447

April, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MILLING METHODS AT THE HUGHESVILLE CONCENTRATOR OF THE ST. JOSEPH LEAD CO., HUGHESVILLE, MONT.

BY

WM. O. VANDERBURG

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April, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MILLING METHODS AT THE HUGHESVILLE CONCENTRATOR OF THE ST. JOSEPH LEAD CO., HUGHESVILLE, MONT.[1]

By Wm. O. Vanderburg[2]

INTRODUCTION

This paper describing the milling methods at the Hughesville concentrator of the St. Joseph Lead Co., Hughesville, Mont., is one of a series of similar papers being prepared by the U. S. Bureau of Mines on practices at the various mills in the United States.

The writer expresses his thanks for the courtesies extended by the officials of the company while visiting the property and acknowledges the assistance and helpful suggestions given by R. H. Willcomb, general manager, and Carl Martin, mill superintendent, of the Hughesville Division of the St. Joseph Lead Co.; and E. D. Gardner, supervising engineer, U. S. Bureau of Mines, Southwest Experiment Station, Tucson, Ariz., in gathering and compiling the data herein presented.

LOCATION

The concentrator is located in Barker Canyon of the Little Belt Mountains, 66 miles southeast of Great Falls, Mont. The mill site is on a hillside about 2 miles from the mine. Ore is transported from the mine to the upper end of the mill by an aerial tramway 10,250 feet long. The upper terminal of the tramway is 5,930 feet and the lower terminal 5,548 feet above sea level. Factors influencing the choice of the mill site were the availability of ground for tailings disposal and the desirability of maintaining a railroad grade under 3 per cent to the mill. A company-owned standard-gage railroad 10.8 miles long connects the mill with the Neihart Branch of the Great Northern Railroad at Monarch.

The mine is in Judith Basin County and the mill in Cascade County, Mont.

GENERAL

The mill has a daily capacity of 400 tons. An average of 25 men is required to operate the mill on three shifts of eight hours each. Two classes of concentrates are made: A lead concentrate which for the year 1929 averaged 61 per cent of lead, 5.7 per cent of zinc, 50 ounces of silver, and 0.05 ounce of gold; a zinc concentrate averaging 51 per cent of zinc, 1 per cent of lead, 31 ounces of silver, and 0.02 ounce of gold.

The ore occurs in a fissure vein in syenite, and it is mined by the horizontal cut-and-fill system. About 35 per cent of the material blasted is rejected underground as

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6447."

2 - Associate mining engineer, U. S. Bureau of Mines.

waste and is used to fill the mined-out portions of the vein. Selective mining and hand-sorting are practiced to a great extent to obtain a higher-grade product for concentration. Hand-sorting is employed both underground and on the surface. About 8 per cent of the run-of-mine ore is rejected as waste by hand-sorting at the grizzly ahead of the primary breaker.

The water supply for the mill is derived from two sources: Dry Fork Belt Creek, and one of its tributaries, known as Gold Run Creek. All the mill water supply during the summer months has so far been taken from Gold Run Creek. During the winter the tributary creek freezes over and the water gets low, so the mill is supplied with water from Dry Fork Belt Creek. The surge tank for the water supply has a capacity of 85,000 gallons. Water flows by gravity from the dam located on Gold Run Creek through 6,000 feet of 5-inch redwood stave pipe. The water from Dry Fork Belt Creek is elevated to the surge tank in one lift by a single-stage centrifugal pump which has a capacity of 350 gallons per minute. The pipe line consists of 2,500 feet of 4-inch and 800 feet of 6-inch standard iron pipe, the latter connecting the surge tank with the mill. By arrangement of valves which control the flow of water into the mill circuit, the 6-inch portion of the line serves the double purpose of a discharge into the surge tank or into the mill circuit, as required. When the mill is using less water than is being pumped the excess water is forced up the 6-inch line to the surge tank; when more water is required the additional water flows from the surge tank through the 6-inch line to the mill. A check valve is located at the end of the 4-inch line which prevents the water from flowing from the surge tank back through the pump. This is necessary because of a small by-pass at the pump which drains it and the 4-inch line as soon as the pump stops. The by-pass drain prevents freezing of the pump during cold weather in the event that the pump stops. The water requirements of the concentrator amount to about 320 gallons per minute. The mill water is not reclaimed.

Electric power is purchased from the Montana Power Co. The power is generated at hydroelectric plants located on the Missouri River, a short distance from Great Falls. It is transmitted to the property at 23,000 volts, and it is stepped down to 440 volts for use in the mill motors of 3-hp. or larger. The small motors use a voltage of 110 or 220.

ORE TREATED

The economic metals that occur in the ore are lead, zinc, silver, and gold. The average grade of the ore treated for the year 1929 was as follows: 6.21 per cent of lead, 5.01 per cent of zinc, and 9.09 ounces of silver per ton. The ore is practically all sulphide in character. The ore minerals are galena, sphalerite, marmatite (ferriferous sphalerite), and pyrite. A small amount of copper occurs in the concentrates as cupriferous pyrite and chalcopyrite. The galena and sphalerite carry most of the silver. The gangue consists principally of altered syenite and altered rhyolite with subordinate amounts of calcite, barite, quartz, rhodocrosite, and marcasite. The moisture content of the ore as delivered to the concentrator averages about 2.5 per cent.

The ore crushes readily. Fine grinding is necessary to liberate the zinc minerals which are more or less interlocked with the pyrite. The galena is coarsely crystalline, and the classifier return to the rod mill shows the greater part of the galena to be free from gangue.

HISTORY OF CONCENTRATOR OPERATIONS

The first concentrator in the district of 75 tons daily capacity was built in 1911,

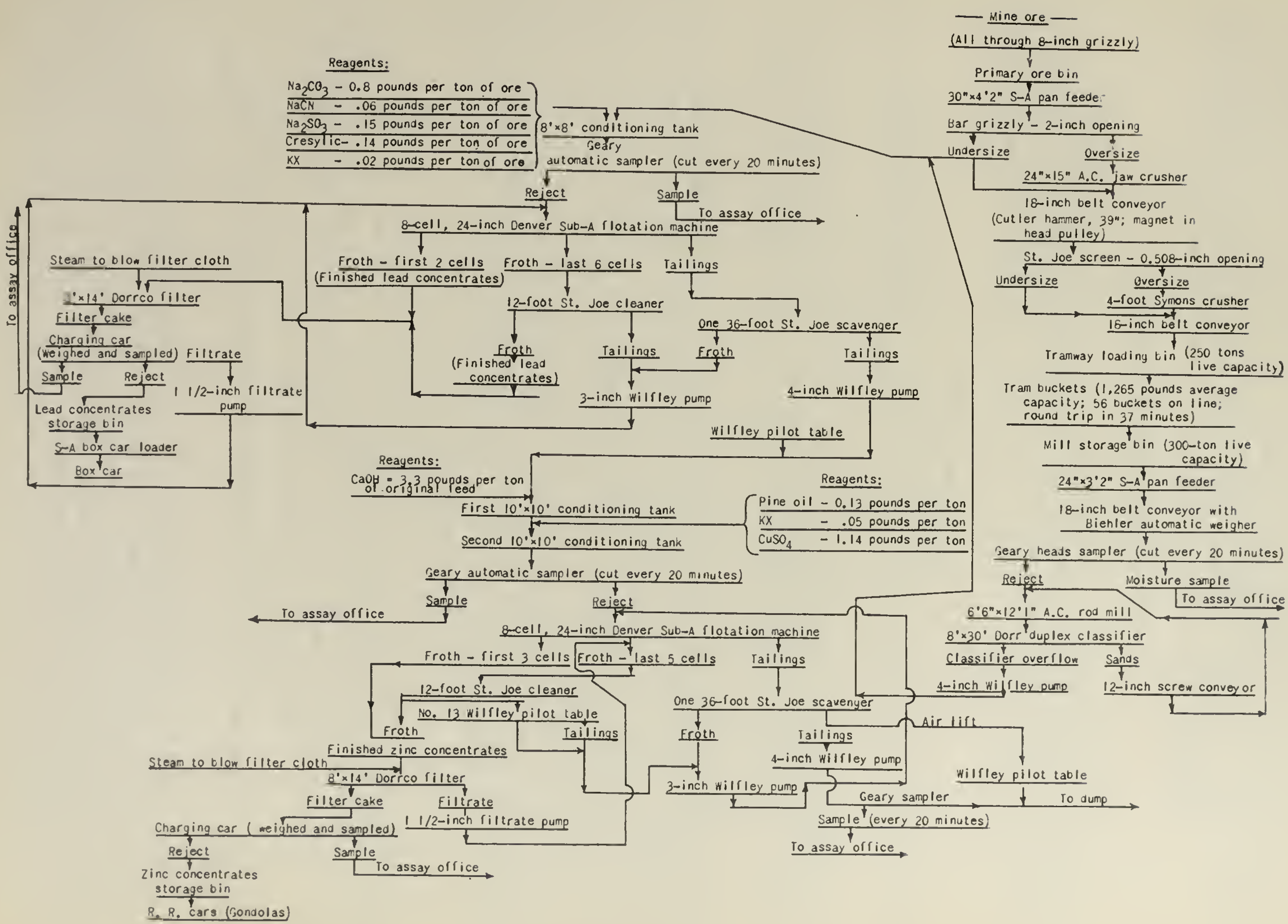

Figure 1.- Flow sheet of crushing plant and mill

and used gravity methods of concentration. In 1911 this mill was dismantled, and in 1920 it was rebuilt and run intermittently until 1927 when the present company acquired control of the principal mines. The present mill of 400 tons daily capacity was designed and built with an all-flotation flow sheet. Building of the mill was started on May 17, 1928, and was completed November 9, 1928. With the exception of several minor mechanical changes, made after the mill was erected, the flow sheet is essentially the same as first designed.

PRESENT METHOD OF CONCENTRATION

The general flow sheet of the concentrator is shown in Figure 1. The method of flotation used is a selective separation of the lead and zinc sulphides in the original pulp. Very little elevating of the ore or pulp is necessary. The average quantity treated per 24 hours in 1929 was about 312 tons.

Breaking and Crushing

Breaking and crushing of the ore is done at the mine. All the ore is broken underground so that it will pass through an 8-inch grizzly.

The mine ore is delivered into a two-compartment bin with inclined bottoms lined with steel which slope two ways. The capacity of each side of the bin is approximately 50 tons. Waste rock is dumped into one side of the bin and ore into the other.

From the bin the ore is delivered to a bar grizzly with 2-inch spaces, by a 30 by 50 inch Stephens-Adamson pan feeder driven by a 3-hp. motor. At the feeder a sorter removes pieces of waste rock from the ore and drops the waste into a bin at his side. Approximately 30 per cent of the grizzly feed passes through as undersize. The oversize product of the grizzly is fed to a 15 by 24 inch Blake-type crusher. The crusher has manganese steel wearing plates and is driven by a 40-hp. motor operated at 720 revolutions per minute. The grizzly undersize and the Blake-crusher product are delivered by gravity to an 18-inch belt conveyor which travels 300 feet per minute. A 39-inch Cutler-Hammer magnet, operating at 17 amperes and 125 volts, is suspended on a head pulley over the conveyor belt to remove pieces of tramp iron. The conveyor belt discharges on a St. Joe vibrating screen with 0.508-inch openings, the screen oversize feeding a 4-foot Symons cone crusher.

The Symons crusher is direct connected to a 100-hp. alternating-current motor making 514 revolutions per minute. The screen undersize and the product from the cone crusher are delivered by gravity to a rubber-surfaced belt conveyor 18 inches wide traveling 301 feet per minute. The belt conveyor discharges into the tramway loading bin which is flat-bottomed and built of wood and has a capacity of 250 tons, "live" load.

Table 1 gives screen sizes of the Symons-cone-crusher feed and the final crushing-plant product.

The aerial tramway which conveys the crushing-plant product to the concentrator storage bin is equipped with stationary track cable, supporting carriers, and has a capacity of 32.5 tons of ore per round trip, which requires 37 minutes. The buckets are attached and detached from the carrier rope automatically, although supervision is required for spacing the buckets at appropriate intervals at both terminals and for loading and dumping the buckets. The traction rope is below the carriers and the grips are of the underhung type.

The mill storage bin has a "live" capacity of 300 tons. Ore is fed from this bin to the grinding mill at the rate of 13 tons per hour by a 24 by 38-inch Stephens-Adamson pan feeder. The feeder discharges onto a rubber surfaced conveyor belt inclined at an angle of 10° and operated at a speed of 134 feet per minute. The ore in traveling on this belt is weighed by a Biehler automatic weigher which registers in units of 100 pounds.

Grinding

The conveyor belt delivers the ore to an Allis-Chalmers rod mill 6 feet 6 inches in diameter and 12 feet 1 inch in length, operated at a speed of 18 revolutions per minute by a 240-hp. alternating-current motor making 1,200 revolutions per minute. The gear reduction is 67 to 1. The rod mill is operated in closed circuit with an 8 by 30 foot Dorr duplex classifier, making 21 strokes per minute. The amount of circulating load is about 300 per cent. Manganese steel liners are used in the rod mill, the liner consumption amounting to 0.247 pound per ton of ore. Rod consumption amounts to 1.986 pounds per ton of ore.

The scoop feeder on the rod mill has a stellite lip about 1/8 inch thick to minimize wear. The stellite is welded to the lip locally by means of an acetylene welding torch.

The classifier overflow which comprises the flotation feed contains 23 per cent of solids and is pumped by a 4-inch Wilfley pump to an 8 by 8 foot wooden conditioning tank equipped with a Denver agitating mechanism.

Table 1 gives screen sizes of rod-mill feed and discharge products, and classifier sand and overflow products.

Flotation Lead Circuit

From the lead-circuit conditioning tank the pulp goes to an 8-cell, 24-inch, Fahrenwald Denver Sub-A flotation machine with the cells arranged in series. The first two cells make finished lead concentrates and the other six cells make middlings and tailings. The middlings go to a 12-foot St. Joe flotation machine operating as a cleaner unit and the tailings to a 36-foot St. Joe machine used as a scavenger. The concentrate from the cleaner is a finished lead product. The tailings from the cleaner are returned to the Fahrenwald cells by-passing the lead-circuit conditioning tank. The froth from the scavenger unit is also returned to the Fahrenwald cells. The tailings from the scavenger cells go to the conditioning tanks at the head of the zinc circuit.

Each of the Fahrenwald cells is equipped with a 5-hp. alternating-current motor connected to the impeller shaft by a Texrope drive and making 1,150 revolutions per minute.

The St. Joe flotation cells operate on the bubble column principle. The air necessary to form the bubble column in the 36-foot scavenger and 12-foot cleaner is supplied by a centrifugal blower direct connected to a 27-hp. alternating-current motor making 3,500 revolutions per minute. The capacity of the blower is 4,200 cubic feet of air per minute under a pressure of 0.8 pound per square inch. A decided advantage in the use of the scavenger machine is its ability to function as a safety valve in the event that something goes wrong in the circuit ahead of the machine. The scavenger machine will continue to discharge a clean tailings product, even if it is heavily overloaded for 20 minutes.

A peculiar condition noted in the wear of flotation equipment is that the impellers

of the lead-circuit flotation cells, the impellers of the conditioning tank, and the aerating pipes in the scavenger cells wear out much faster than similar equipment of the zinc circuit. The impellers of the lead flotation cells handled 110,350 tons of ore while the impellers of the zinc flotation cells up to June 1, 1930, had handled 160,500 tons and appear to be good for about 40,000 tons more.

The selective problem presented in the flotation treatment of this ore is three-fold; the separation of the lead and zinc minerals, the recovery of the cupriferous pyrite with its accompanying silver, and the rejection of the marcasite and most of the pyrite which is low in silver.

The amounts and kind of reagents added to the pulp in the conditioning tank at the head of the lead circuit are as follows:

Reagent	Pounds per ton of ore treated
Soda ash	1.466
Sodium cyanide	0.0616
Sodium sulphite	0.1818
Cresylic acid	0.1826
Potassium xanthate	0.0246

No reagents are added to the lead circuit in addition to those introduced in the conditioner as the amount of reagents added to the pulp in the conditioning tank is sufficient to complete the cleaning operation.

Soda ash is added as the alkaline reagent. It improves the froth and tends to make cleaner concentrates.

Sodium cyanide is used as a depressant for the marcasite and pyrite.

Sodium sulphite is used as a zinc depressant. An excess of this reagent in the circuit has no appreciable effect on the appearance of the froth or on the recovery of the lead.

Cresylic acid is used as a frothing agent. The appearance of the froth changes with the lead content of the heads, and the amount of cresylic acid is increased or decreased to take care of this variation.

Potassium xanthate is used principally as a collector.

When the mill was first started zinc sulphate and sodium cyanide were used to depress the zinc in the lead circuit. The change from zinc sulphate to sodium sulphite resulted in lowering the zinc content of the lead concentrates from 6-1/2 to 7 per cent down to 4 to 4-1/2 per cent. Lead concentrates containing about 62 per cent of lead have been found to yield the maximum economic recovery of silver in the lead concentrates.

Table 1 gives the screen sizes of heads to the lead flotation circuit and also screen sizes of the lead concentrates. Table 2 gives a screen-assay analysis of feed to the flotation lead circuit. Table 3 gives the lead, zinc, and iron contents of the individual rougher-cell concentrates, the cleaner-unit products, and the scavenger-machine feed and tailing products of the lead circuit.

Flotation Zinc Circuit

The tailings product from the lead scavenger unit which is the feed to the zinc flotation circuit goes to two wooden conditioning tanks operating in series, each 10 feet in diameter, 10 feet high, and each equipped with a Denver agitating device. In the first conditioner lime is added, and in the second pine oil, potassium xanthate, and copper sulphate.

The flotation cells of the zinc circuit are identical with those of the lead circuit.

From the second conditioning tank the pulp passes to a Fahrenwald machine with 8 cells arranged in series. These cells make three products: Finished concentrates from the first three cells, middlings from the last five cells, and tailings. The middlings froth goes to a 12-foot St. Joe cleaner which makes finished concentrates and tailings. The tailings are returned to the first Fahrenwald cell. The tailings from the Fahrenwald cells go to a 36-foot St. Joe scavenger machine which makes a middlings froth which is returned to the head of the Fahrenwald cells and tailings which go to waste.

The only dilution of the pulp in the zinc circuit is from the water added to the concentrates launders and returned to the circuit by the filtrate pump. Diluting water is kept as low as possible. The amount and kind of reagents added to the pulp in the conditioning tanks ahead of the zinc circuit are as follows:

Reagent	Pounds per ton of ore treated
First conditioning tank:	
Lime	3.337
Second conditioning tank:	
Pine oil	0.125
Potassium xanthate	0.0399
Copper sulphate	1.210

Lime is used as a depressant for the marcasite and pyrite.
Pine oil is used as the frothing agent.

Potassium xanthate is used as a froth stiffener and as a collector. In the zinc circuit the xanthate is very sensitive; a deficiency increases the zinc content of the tailings and an excess produces concentrates of high iron content. An excess of pine oil will also increase the amount of iron in the concentrates. An increase in the amount of xanthate added necessitates an increase in the amount of pine oil.

Copper sulphate is the reactivating agent for the zinc minerals which have been temporarily depressed in the lead circuit.

Table 1 gives screen sizes of the zinc-flotation concentrates and the final concentrator tailings.

Table 2 gives a screen-assay analysis of the final concentrator tailings.

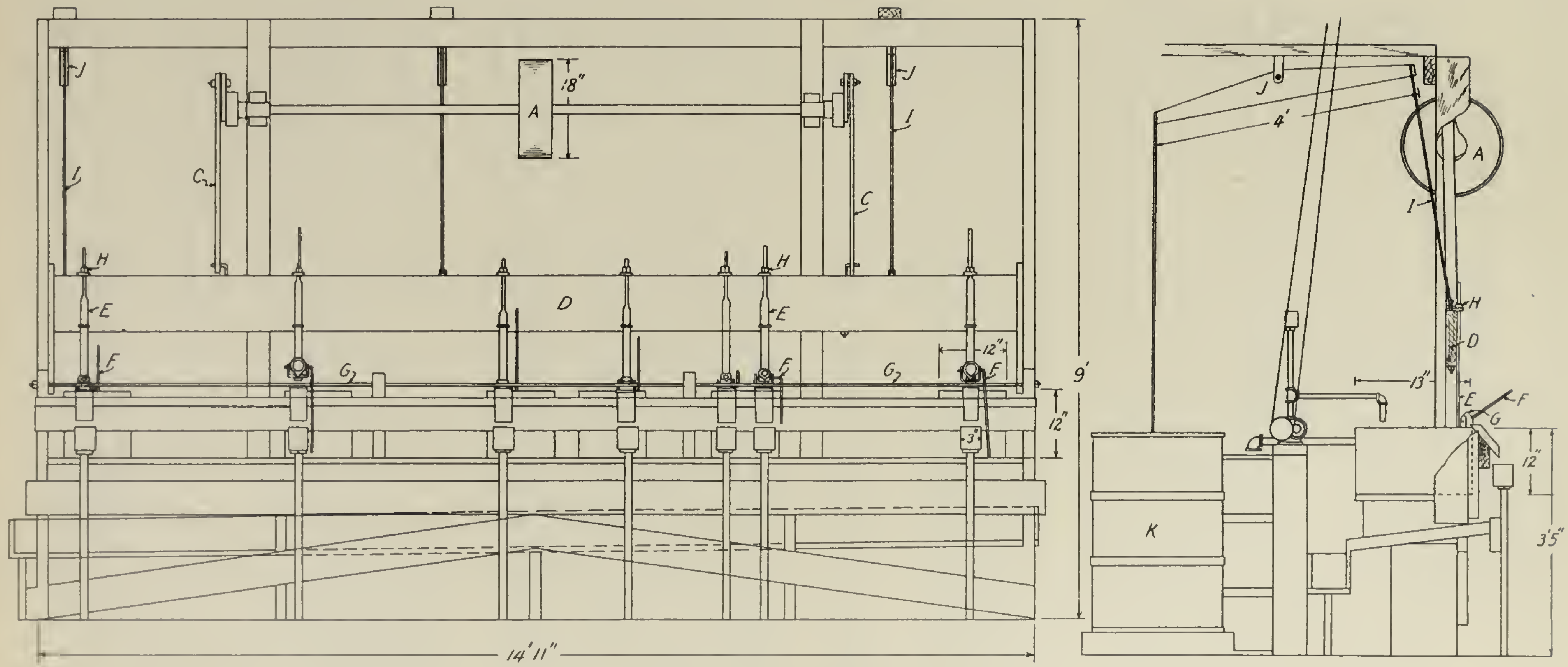

Figure 2.- Reagent feeder

Table 3 gives the lead, zinc, and iron contents of the individual rougher-cell concentrates, the cleaner-unit products, and the scavenger-machine products of the zinc section.

Conditioning of Pulp and Feeding of Reagents

All the reagents used are added to the conditioning tanks as solutions, except the soda ash added to the lead circuit and the lime added to the zinc circuit. The soda ash and lime are fed uniformly at the rate of 19 and 43-1/2 pounds per hour, respectively, by conveyor-belt feeders each driven by a 1/4-hp. motor. The rate at which the reagents are fed from hoppers onto the conveyor belts is controlled by a cone drive and a gate at the discharge end of the hoppers. All reagents for the flotation of lead, except cyanide, are added to the lead conditioner through a stationary column 12 inches in diameter and 5 feet 4 inches long. The feed to the conditioner also enters through this column. The cyanide is added to the conditioner outside the column in order that the sodium sulphite will have the first action on the sphalerite as cyanide has a tendency to float sphalerite. By omitting the cyanide entirely, lead concentrates assaying about 45 per cent of lead and 2.7 per cent of zinc are produced. The time of conditioning pulp for the lead circuit is 5 minutes.

As previously noted, two conditioning tanks, operating in series, are used in the zinc circuit. With these conditioners in series, sufficient time is given for the conditioning of the pulp, and the use of an excessively large unit is avoided. The conditioning time for the pulp of the zinc circuit is 30 minutes. The impeller in each conditioning tank is driven by a Texrope drive from a 5-hp. motor.

All liquid reagents, except copper sulphate, are fed by the reagent feeder shown in Figures 2 and 3.

The operation and control of this reagent feeder are as follows: Referring to Figure 2, the pulley A on the drive shaft B actuates the rods C, which are eccentrically connected to either end of the drive shaft. The length of the eccentric arms at the ends of the drive shaft and the ratio of the motor and drive shaft pulleys are such that the rods C lift the board D to which they are attached about 10 inches, 16 times per minute. To the board D are fastened the straps E. At the lower end of each strap a small iron cup is pivoted as shown in greater detail in Figure 3. To each cup is secured a small iron rod F which slides upon the rod G as the cup is raised and lowered. The angle of tip of each cup determines the quantity of solution emptied into the launder each time the cup is raised. The angle of tip is adjusted by the nut R attached to each strap.

The rods I of Figure 2 are attached to the ends of the board D and the rocker arms J. To the other end of each rocker arm a rope is attached with a perforated pail tied to the free end of each rope. As the board D moves up and down the rocker arms move the pails which are suspended in the mixing barrels K. The mixing device serves to mix thoroughly the sodium cyanide, potassium xanthate, and sodium sulphite solutions before they are pumped to the reagent feeder by 1/2-inch rotary pumps.

The control of the amount of reagents is easily adjustable, and the device is accurate and reliable. The feeder is driven by a 3-hp. motor.

The feeding of the copper sulphate as shown in Figure 4 is done from a wooden barrel, which contains the solid salt and through which a controlled volume of water flows giving a saturated solution at all times. The water feed to the barrel is controlled by the reagent feeder of Figures 2 and 3.

This method of feeding copper sulphate has been found satisfactory, and due to the even temperature maintained in the mill by the heating system there is little tendency for the saturated solution to crystallize before reaching the conditioning tank.

All reagents are added to the conditioning tanks from the central floor, thus giving control of the entire plant from one point.

SAMPLING AND MILL CONTROL

Samples of the rod-mill feed, the heads of the lead and zinc circuits, and the mill tailings from the zinc-scavenger unit are taken by Geary samplers operated by an electrically controlled mechanism. The rod-mill sample is taken principally for the determination of moisture in the ore, although it is also used as a check on the heads sample. The sample cuts are taken simultaneously every 20 minutes, by all four samplers. The samples thus taken are combined into four composite samples for each shift and assayed for lead, zinc, and iron. The mill-tailings samples are not assayed for iron.

Pipe samples are taken of the lead and zinc concentrates. After weighing each hopper load of concentrates the weigher takes a pipe sample. A hopper load of lead concentrates weighs about 4,000 pounds, and one of the zinc concentrates weighs about 3,000 pounds. The pipe samples are combined into composite samples for each class of concentrates for each shift. The zinc-concentrates composite samples are assayed for insoluble matter as well as lead, zinc, and iron.

In addition to the samples mentioned the mill operations are controlled by the use of three 18 by 40 inch Wilfley pilot tables, which continuously treat a portion of the tailings from the lead-scavenger cells, the concentrates from the zinc-cleaner cells, and the mill tailings from the zinc-scavenger unit.

Each pilot table is driven by a 1/4-hp. motor. The use of the small concentrating tables as continuous vanning plaques furnishes a continuous visual check on the mill operations.

The standard La Motte comparator is used for the determination of alkalinity - or, more accurately, for the determination of pH values. Thymol blue is used in the lead circuit and bromothymol blue for the mill water. The pH value of the mill water is 7.6, that of the pulp in the conditioner ahead of the lead circuit is 8.4, and the pH value of the tailings pulp is approximately 11.0.

The control of the lead and zinc circuits is not restricted to pH readings but is based largely on the appearances of the froths, and the use of vanning plaques. The flotation operators have become proficient in handling the circuits by visual inspection, and the changing of the amounts of reagents is largely left to their judgment.

FILTERING

Finished concentrates products are produced in the lead circuit by the first two Fahrenwald cells and the St. Joe cleaner unit.

On the zinc side the first three Fahrenwald cells and the St. Joe cleaner unit produce finished concentrates.

In both the lead and zinc circuits the finished concentrates are sent to a Dorrco filter, 8 feet in diameter and 14 feet long driven by a 5-hp. motor. The material used for the filtering medium is Palma twill, style 31, made by the Filter Fabrics Corporation. Each filter has 336 square feet of filtering area. On the lead filter the average life of the filter cloth is about five weeks and on the zinc side about three weeks. The shorter life of the cloth on the zinc side is due to the fact that the cloth blinds more rapidly on the zinc side.

The filter cake is dewatered by suction before it reaches the discharge point at the top of the drum. A pulsating valve at the top of the drum automatically reverses from suction to compression three times while the cake passes the top section. The pulsations discharge the cake onto a conveyor belt. Constant use of the filter tends to clog the pores in the cloth with slimes. This is overcome by using live steam an average of about an hour each day. The steam is admitted through the blow connections and the blow is cut off at this time. The steam tends to loosen the pores in the filter cloth and to force out the slimes that have collected in them.

The vacuum for each filter, equal to 21 inches of mercury, is supplied by an Ingersoll Rand, 22 by 8 inch, belt-driven vacuum pump. The pressure used for freeing the filter cake from the cloth is 5 pounds per square inch. Each vacuum pump is driven by a 30-hp. motor operating at 1,165 revolutions per minute.

The moisture content of the lead and zinc filter cakes is 7 and 9 per cent, respectively.

The filtrate on the lead side is returned to the head of the Fahrenwald cells, and the filtrate on the zinc side is returned to the fourth Fahrenwald cell.

WEIGHING AND LOADING OF CONCENTRATES

The lead and the zinc concentrates are conveyed by the traveling belt inside of each filter drum to the weighing hoppers. Each hopper is connected to Howe scales for weighing the concentrates before shipment. Each hopper is equipped with a 1/4-inch sheet-iron partition as shown in Figure 5 so that when the hopper is dumped at either side, the concentrates fall by gravity to either of two bins for each class of concentrates. Each bin compartment has an inclined bottom lined with sheet iron, and will hold 105,000 pounds of zinc concentrates or 88,000 pounds of lead concentrates.

The zinc concentrates are shipped in bottom-dump, steel railroad cars, and the lead concentrates in box cars. The box cars are used so that they may be utilized for a return haul with mill supplies. The box cars are loaded by means of a Stephens-Adamson box-car loader. With this loader the lead concentrates are run by gravity into the hopper on the machine and are thrown tangentially off an endless belt driven by a small motor and Texrope drive. The carrying surface of the belt travels in a concave curve over rollers, and as the load on the belt travels around the curve formed by the belt, the speed of travel throws the material to the end of the car. The discharge angle of the belt can be varied to pile the material to any height. A box car can be loaded with this machine in about two hours.

TAILINGS DISPOSAL

The tailings are pumped to a pond through 1,400 feet of 4-inch standard iron pipe

by a 4-inch centrifugal pump direct connected to a 30-hp. motor making 1,160 revolutions per minute. The delivery pipe is laid on oil barrels or 2 by 6 inch crossed timbers, along the crest of the tailings pond which is located in a shallow swale. A T-fitting is placed at the junction of each 16-foot length of pipe. In the open end of each T-fitting a 4 by 1 inch reducer is fitted, through which the tailings are diverted into the pond. By plugging the 1-inch holes and by means of valves placed at intervals along the delivery line, the tail ings can be directed to any portion of the pond. The crest of the pond is built up continuously with tailings shoveled by hand. The excess water is drained off through box launders underneath the pond.

LABOR

The mill labor is composed entirely of American-born men, who are efficient and steady. All mill work is done on company account and the wage scale for the principal classifications of employees as of April, 1930, was as follows:

Crusherman	$5.50
Rod-mill operator	5.00
Flotation operator	5.50
Filterman	4.50
Sampler and reagent mixer	5.00
Loader	4.50
Tramway operator	4.50 to 5.00
Tailings-pond man	4.50

TABLES

Table 1 gives screen sizes of Symons cone-crusher feed, concentrator feed, intermediate and final concentrator products. Table 2 gives screen-assay analyses of flotation feed and flotation-tailings products. Table 3 gives the lead, zinc and iron contents of concentrates produced by the individual rougher cells and also the lead, zinc, and iron contents of cleaner and scavenger unit products for both the lead and zinc circuits.

Table 4 shows metallurgical data for the year 1929. Table 5 gives average assays of concentrator heads and final products and the percentage distribution of the silver, lead, and zinc in concentrator final products for the year 1929. Table 6 shows cost summaries in units of labor, power and supplies for the year 1929.

Table 1.- Screen sizes of crushing-plant and concentrator products

Screen size	Symons cone-crusher feed		Rod-mill feed		Classifier feed		Classifier sands		Classifier overflow		Lead concentrates		Zinc concentrates		Tailings	
	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent	Per cent	Cumulative per cent
On 3-inch ring	3.6	3.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-
On 2 1/2-inch ring	7.7	11.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-
On 2-inch ring	9.0	20.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-
On 1-inch square	26.2	46.5	0.7	0.7	-	-	-	-	-	-	-	-	-	-	-	-
On 1/2-inch square	-	-	12.1	12.8	-	-	-	-	-	-	-	-	-	-	-	-
On 3 mesh	23.3	69.8	37.2	50.0	-	-	-	-	-	-	-	-	-	-	-	-
On 4 mesh	3.4	73.2	10.1	60.1	-	-	-	-	-	-	-	-	-	-	-	-
On 6 mesh	3.7	76.9	6.8	66.9	-	-	-	-	-	-	-	-	-	-	-	-
On 10 mesh	6.9	83.8	11.1	78.0	-	-	-	-	-	-	-	-	-	-	-	-
On 14 mesh	3.3	87.1	4.7	82.7	-	-	-	-	-	-	-	-	-	-	-	-
On 20 mesh	2.9	90.0	3.8	86.5	0.4	0.4	0.6	0.6	-	-	-	-	-	-	-	-
On 28 mesh	1.9	91.9	2.7	89.2	0.9	1.3	1.1	1.7	-	-	-	-	-	-	-	-
On 35 mesh	2.4	94.3	3.1	92.3	3.1	4.4	4.3	6.0	-	-	-	-	-	-	2.2	2.2
On 48 mesh	1.8	96.1	2.3	94.6	10.5	14.9	12.9	18.9	1.1	1.1	-	-	-	-	11.7	13.9
On 65 mesh	1.3	97.4	1.6	96.2	16.1	31.0	17.9	36.8	9.0	10.1	0.2	0.2	0.8	0.8	4.9	18.8
On 80 mesh	0.3	97.7	0.4	96.6	6.8	37.8	7.7	44.5	4.4	14.5	-	-	1.0	1.8	11.2	30.0
On 100 mesh	0.6	98.3	0.8	97.4	14.8	52.6	16.2	60.7	10.3	24.8	0.8	1.0	4.8	6.6	14.8	44.8
On 150 mesh	-	-	-	-	18.1	70.7	20.3	81.0	14.1	38.9	2.9	3.9	14.9	21.5	13.8	58.6
On 200 mesh	-	-	-	-	10.5	81.2	10.2	91.2	13.6	52.5	7.1	11.0	18.7	40.2	-	-
On 250 mesh	-	-	-	-	3.8	85.0	2.4	93.6	-	-	4.4	15.4	7.6	47.8	-	-
Through 250 mesh	-	-	-	-	15.0	-	6.4	-	-	-	84.6	-	52.2	-	-	-

Table 2.-- Screen-assay analyses of typical flotation feed and tailings

FLOTATION FEED TO LEAD CIRCUIT

Screen size, mesh	Weight, pounds in 100	Analyses, per cent		Weight of metals, pounds		Per cent of total	
		Lead	Zinc	Lead	Zinc	Lead	Zinc
Composite feed..	100.0	6.90	3.79	6.899	3.790	100.0	100.0
On 48	1.1	0.79	0.56	0.009	0.006	0.1	0.2
On 65...	9.0	1.63	1.12	0.147	0.101	2.1	2.6
On 80....................	4.4	1.29	1.28	0.057	0.056	0.8	1.4
On 100	10.3	1.60	2.50	0.165	0.258	2.4	6.8
On 150.............	14.1	1.84	4.75	0.259	0.670	3.8	17.8
On 200.	13.6	3.50	5.00	0.476	0.680	6.9	17.9
Through 200	47.5	12.18	4.25	5.786	2.019	83.9	53.3

FLOTATION TAILINGS

Screen size, mesh	Weight, pounds in 100	Analyses, per cent		Weight of metals, pounds		Per cent of total	
		Lead	Zinc	Lead	Zinc	Lead	Zinc
Composite tailings. ..	100.0	0.31	0.34	0.314	0.343	100.0	100.0
On 48..	2.2	0.16	0.22	0.004	0.005	1.3	1.5
On 65	11.7	0.27	0.40	0.032	0.047	10.2	13.7
On 80.	4.9	0.30	0.48	0.015	0.024	4.8	7.0
On 100.................	11.2	0.31	0.28	0.035	0.031	11.1	9.1
On 150	14.8	0.27	0.48	0.040	0.071	12.7	20.8
On 200	13.8	0.25	0.30	0.035	0.041	11.2	11.7
Through 200	41.4	0.37	0.30	0.153	0.124	48.7	36.2

Table 3.--Analyses of flotation products under normal operating conditions

Product	Analyses, per cent		
	Lead	Zinc	Iron
No. 1 lead-cell concentrates	72.8	2.9	3.8
No. 2 lead-cell concentrates	68.2	3.75	5.1
No. 3 lead-cell concentrates	51.0	7.1	10.7
No. 4 lead-cell concentrates	37.6	8.8	14.8
No. 5 lead-cell concentrates	20.6	10.55	21.3
No. 6 lead-cell concentrates	14.0	11.25	24.0
No. 7 lead-cell concentrates	10.2	11.65	24.7
No. 8 lead-cell concentrates	9.0	11.6	24.2
Lead-scavenger feed......	0.50	4.2	11.3
Lead-scavenger tailings.........	0.35	4.05	10.5
Lead-cleaner concentrates......	59.8	7.10	7.3
Lead-cleaner tailings	8.08	13.55	18.5
No. 1 zinc-cell concentrates	1.10	50.8	6.0
No. 2 zinc-cell concentrates	1.29	50.8	6.2
No. 3 zinc-cell concentrates	1.47	48.2	7.0
No. 4 zinc-cell concentrates	1.65	36.0	10.0
No. 5 zinc-cell concentrates	1.94	38.8	9.8
No. 6 zinc-cell concentrates	2.77	40.4	9.0
No. 7 zinc-cell concentrates	2.57	40.0	10.8
No. 8 zinc-cell concentrates	2.84	28.1	11.4
Zinc-scavenger feed..........	0.20	0.7	-
Zinc-scavenger tailings.........	0.14	0.38	-
Zinc-scavenger concentrates..	0.57	3.8	-
Zinc-cleaner concentrates......	1.86	50.2	6.8
Zinc-cleaner tailings	2.2	9.4	-

Table 4.-- Metallurgical data for the year 1929

Total ore treated, dry tons	106,537
Moisture in ore to mill, per cent	2.51
Hours operated per day	24
Days operated	341.52
Average ore treated per 24 hours, tons	311.85
Total concentrates produced, dry tons	18,762.22
Average lead concentrates produced per 24 hours, dry tons	30.296
Average zinc concentrates produced per 24 hours, dry tons	24.641
Per cent of total lead in lead concentrates	95.46
Per cent of total lead in zinc concentrates	1.48
Per cent of total lead floated	96.94
Per cent of total zinc in zinc concentrates	80.76
Per cent of total zinc in lead concentrates	11.08
Per cent of total zinc floated	91.84
Per cent of total silver in lead concentrates	53.61
Per cent of total silver in zinc concentrates	26.81
Per cent of total silver floated	80.42
Ratio of concentration, lead section	10.29
Ratio of concentration, zinc section	12.65
Net water consumption per ton of ore, tons	6
Average solids in lead-circuit pulp, per cent	23
Average time of conditioning in lead circuit, minutes	5
Average time of conditioning in zinc circuit, minutes	30
Average temperature of lead-circuit pulp, degrees F	35
Average temperature of zinc-circuit pulp, degrees F	40
Rod consumption per ton of ore, pounds	1.986
Liner consumption per ton of ore, pounds	0.247

Table 5.--Average assays and distributions of silver, lead and zinc in final concentrator products for year 1929

Product	Assays				Per cent of total metals		
	Gold, ounces per ton	Silver, ounces per ton	Lead, per cent	Zinc, per cent	Silver	Lead	Zinc
Heads	-	9.09	6.21	5.01	100.00	100.00	100.00
Lead concentrates	0.049	50.19	61.06	5.71	53.61	95.46	11.08
Zinc concentrates	0.022	30.86	1.17	51.20	26.81	1.48	80.76
Tailings	-	2.15	0.23	0.49	19.58	3.06	8.16

Table 6.--Summary of costs in units of labor, power and supplies for year 1929

Tons of dry ore treated: 106,537.

Dry tons of concentrate produced: 18,762.22.

Labor (tons per man per 8-hour shift)	
Crushing and sorting	78.789
Grinding	104.141
Flotation	104.141
Filtering	104.141
Weighing and loading	156.212
Sampling and mixing reagents	312.425
Maintenance	156.212
Supervision	312.425
Assaying	156.212
Warehouse	312.425
Tailings pond	156.212
Power (kw.h. per ton of ore treated)	
Crushing	2.682
Grinding	11.174
Flotation	11.514
Filtering	4.111
Miscellaneous	1.609
Total	31.090
Reagents (pounds per ton of original ore treated)	
Sodium carbonate	1.466
Sodium cyanide	0.0616
Sodium sulphite	0.1818
Potassium xanthate	0.0754
Cresylic acid	0.1826
Pine oil	0.1250
Lime	3.337
Copper sulphate	1.210
Miscellaneous	
Rods (pounds per ton)	1.986
Liners (pound per ton)	0.247

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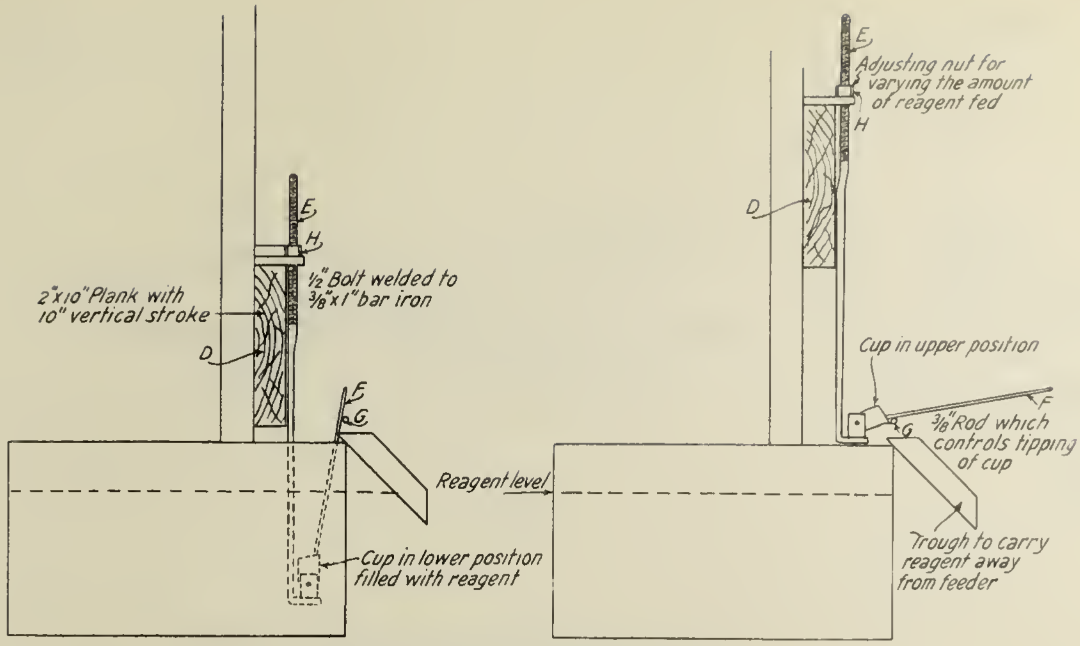

Figure 3.-Details of reagent feeder

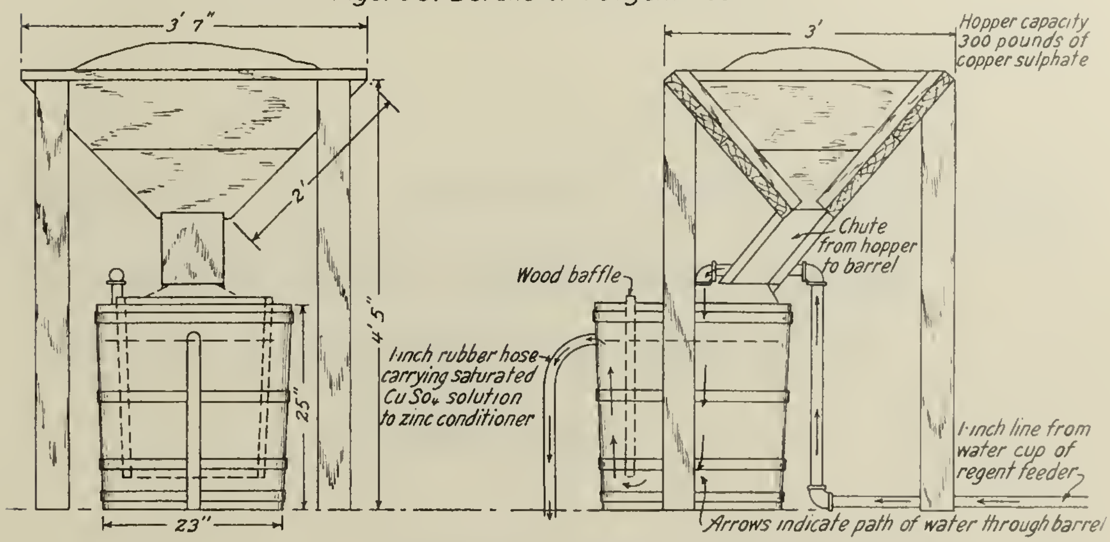

Figure 4.- Copper-sulphate feeder

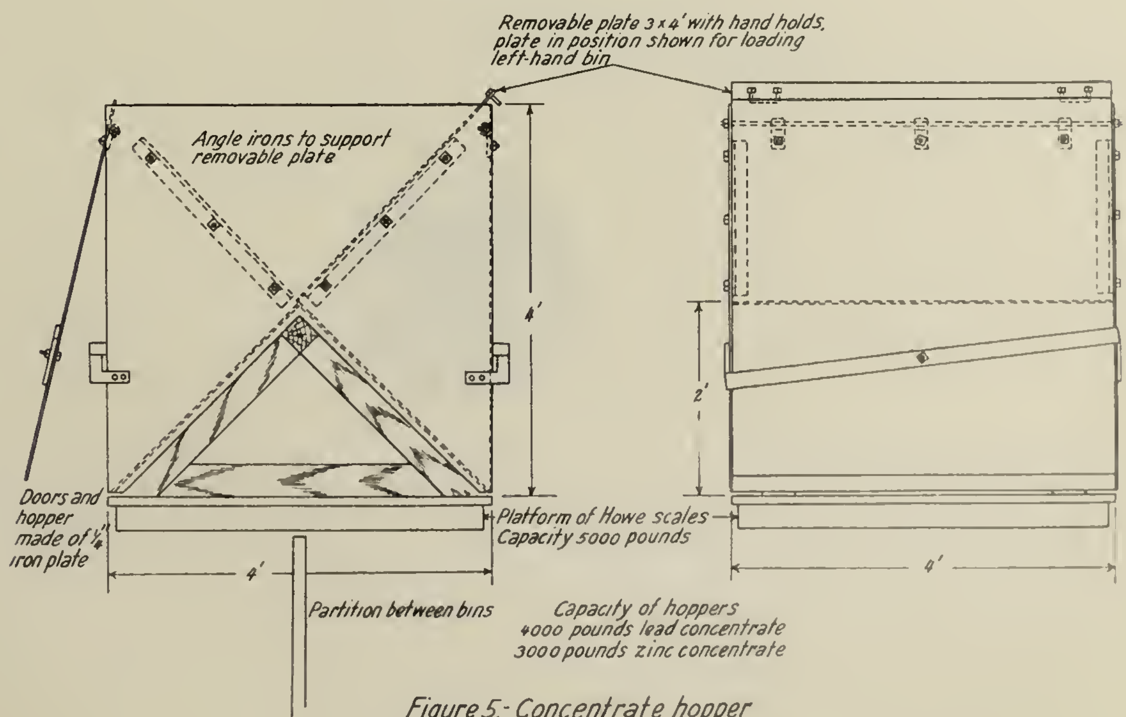

Figure 5.- Concentrate hopper

I. C. 6448 April. 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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MINING, CRUSHING, AND GRINDING METHODS AND COSTS AT THE RELIANCE CEMENT ROCK QUARRY OF THE GIANT PORTLAND CEMENT CO., EGYPT, PA.

BY

S. G. MCANALLY

I.C.6448
April, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING, CRUSHING, AND GRINDING METHODS AND COSTS AT THE RELIANCE CEMENT-ROCK QUARRY OF THE GIANT PORTLAND CEMENT CO., EGYPT, PA.[1]

By S. G. McAnally[2]

INTRODUCTION

This is one of a series of papers describing mining and crushing methods and costs at cement plant quarries throughout the United States and deals directly with the methods employed and costs obtained at the Reliance quarry of the Giant Portland Cement Co. at Egypt, Pa., although it is more or less descriptive of the methods used throughout the Lehigh Valley cement district. Systems of transportation vary according to local conditions, such as proximity to the mill, depth of quarry floor with relation to the mill level, etc. Methods of prospecting differ mainly in the type of drills used. Systems of storage, crushing, and grinding vary chiefly in the types of units.

ACKNOWLEDGMENTS

Most of the information regarding the early history of the Lehigh cement district was obtained from the History of the Portland Cement Industry of the United States, by Robert W. Lesley, the first president of the Portland Cement Association. The writer is indebted also to Charles Clader, Giant Portland Cement Co. quarry superintendent, for valuable information regarding present and past methods and equipment.

Some information on the geology of the district has been gleaned from Prof. Benjamin L. Miller's report on the limestones of Pennsylvania. Other statements regarding the geology are based on personal observation and on the interpretation of the analyses of samples of rock from numerous test holes drilled over a large area.

HISTORY

The Lehigh Valley cement district was discovered and brought into being through the need of cement mortar in the construction of artificial waterways before the advent of the railroads.

About 1830 a small cement plant was built on the Lehigh Canal at Seigfried's Bridge, under the ownership of Gen. J. K. Seigfried, one of the pioneers of the industry. The natural cement manufactured was used largely in the construction of the Lehigh Coal and Navigation Co.'s canal from the coal regions to Easton, Pa. Other cement plants followed and the success of the natural-cement works in the Lehigh district led to the establishment, in the eighties, of a plant at Egypt, Pa.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6448."

2 One of the consulting engineers, U. S. Bureau of Mines and chief chemist, Giant Portland Cement Co.

The Giant Portland Cement Co., originally the American Cement Co., was organized in 1883. In 1898 the company was operating four plants in the vicinity of Egypt. The kilns used were the upright, intermittent type, but in the latter year rotary kilns 60 feet long were installed at one of the above plants. In 1900 the Central mill was built and was equipped with rotary kilns only. The Reliance mill was built in 1905 and the present quarry was opened in the same year.

GEOLOGY

The cement rock of the Lehigh district, as found in the Reliance quarry, belongs to the Jacksonburg formation and is a continuation of a belt that extends from Belvedere, N. J., on the Delaware River, through Stockerton, Bath, and Northampton, to about 10 miles west of the Lehigh River. The belt is only a few miles wide, yet there are about 25 cement plants located on it.

The strata consist of a basal layer of crystalline limestone mixed with dolomite, and an upper layer of argillaceous limestone (cement rock) which comprises most of the formation. The northern boundary of the cement rock can sometimes be determined by an abrupt change in the topography, the line of contact being at the base of the steep slopes which mark the southern margin of the slate belt (Martinsburg formation) which overlies the Jacksonburg limestone. The southern boundary is sometimes marked by a change in slope to the more soluble underlying high-grade limestone (Beekmantown formation), but the geology is not regular.

In some areas large lenses of high-grade cement rock are embedded in very low-grade material; sometimes the rock is covered by residual clay to a depth of 50 feet. There are several large outcrops of dolomite which are probably overlain with cement rock.

The cement rock is a black slaty stone, intermediate in composition between limestone and slate. The calcium carbonate varies from 55 to 85 per cent. In the developed areas the variation is not so extreme and the rock will average about 75 per cent calcium carbonate. This is the approximate percentage required in the raw mix for the manufacture of Portland cement; hence the name "cement rock." The rock is comparatively soft and is laminated due to its slaty nature. Following is the analysis of the average cement rock from the Reliance quarry:

	Per cent
Silica	14.80
Iron oxide and alumina ..	7.20
Calcium carbonate	72.50
Magnesium carbonate	4.10
Water, alkalies, etc. ...	1.40

The above material must be mixed with some high-grade limestone in order to increase the calcium carbonate content to the desired percentage (about 75 per cent). Some cement companies in the Lehigh Valley are more favored than others in having, adjacent to their plants, deposits of limestone which can be quarried and delivered to the mill for the same cost as the cement rock. At some mills the cement rock is sufficiently high in calcium carbonate to require the addition of clay or other argillaceous material in order to regulate the mix. No workable deposit of high-grade calcium limestone has been discovered in the

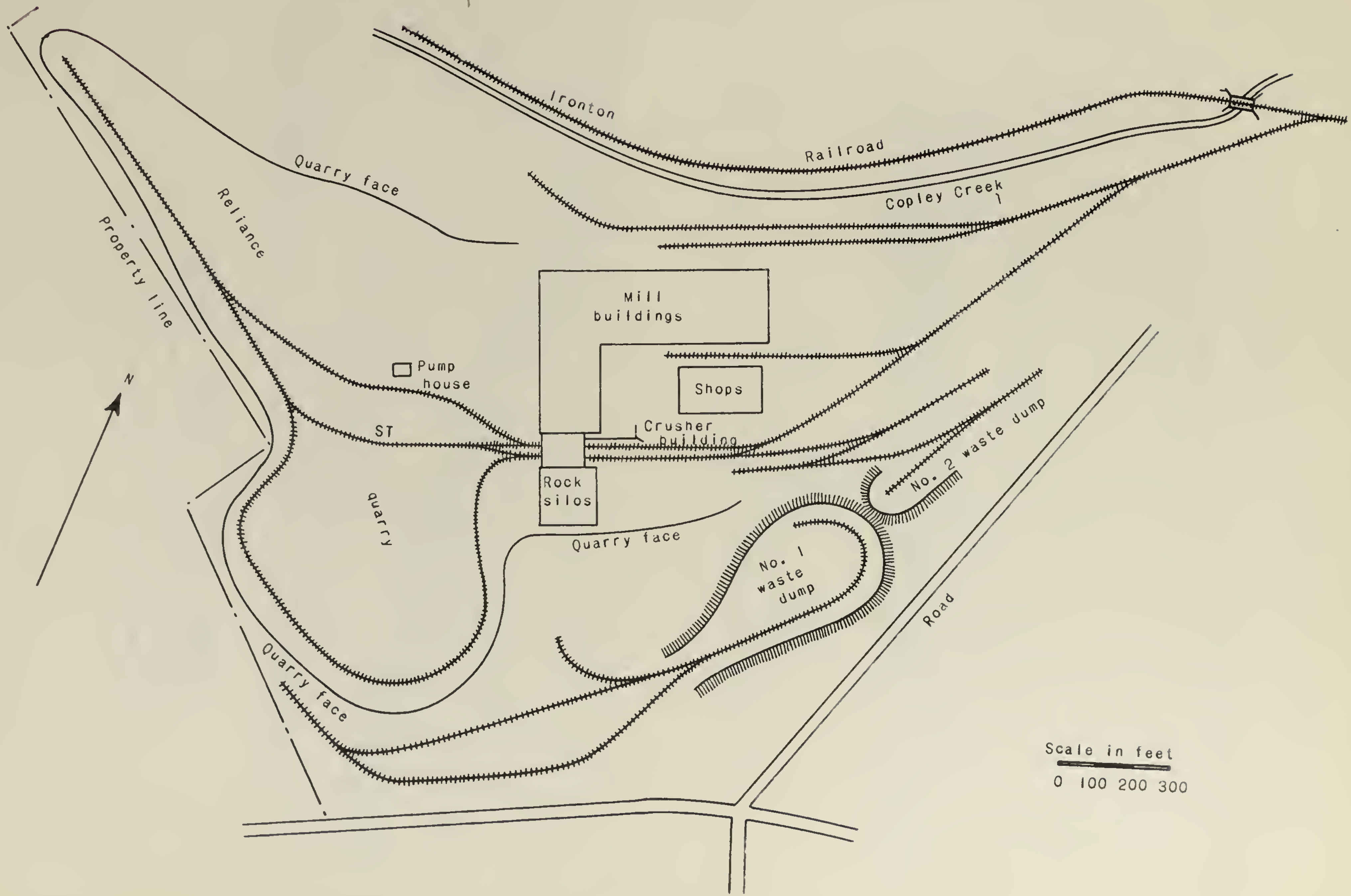


Figure 1.- Plan of Reliance quarry

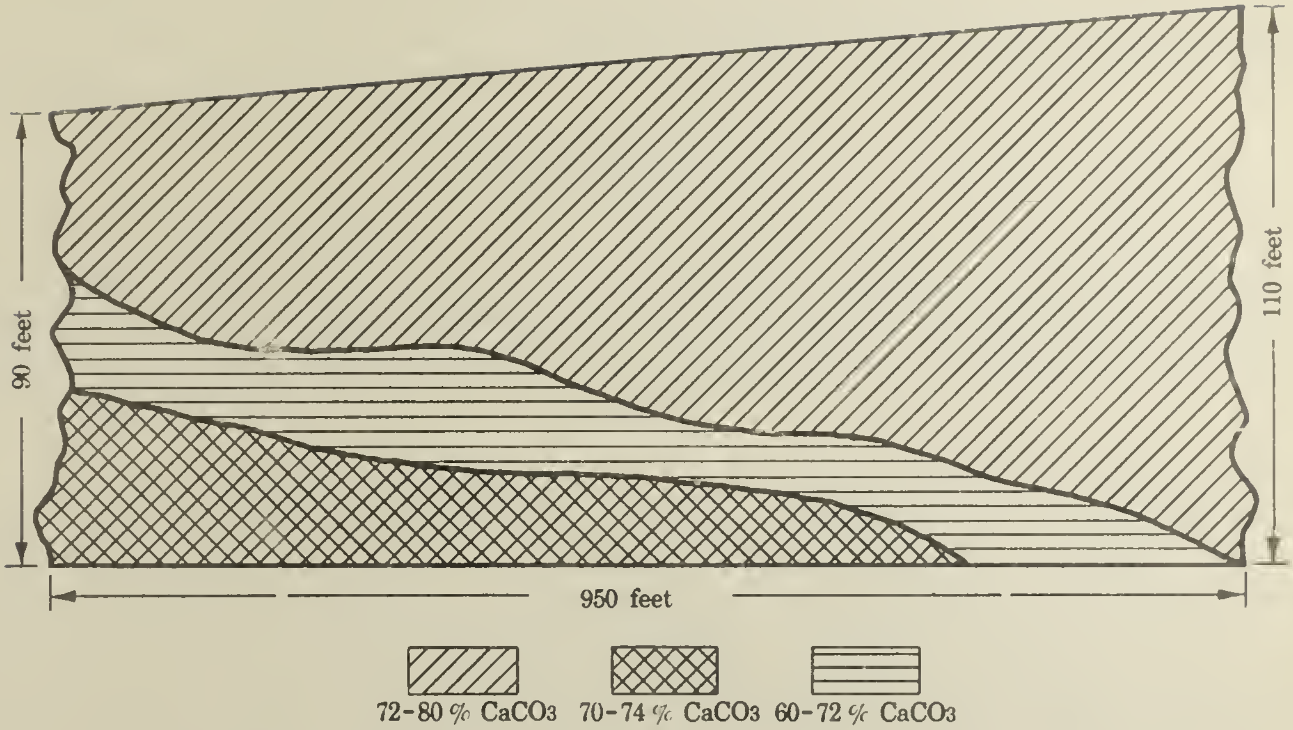


Figure 2. – Section of south face of quarry showing composition of strata

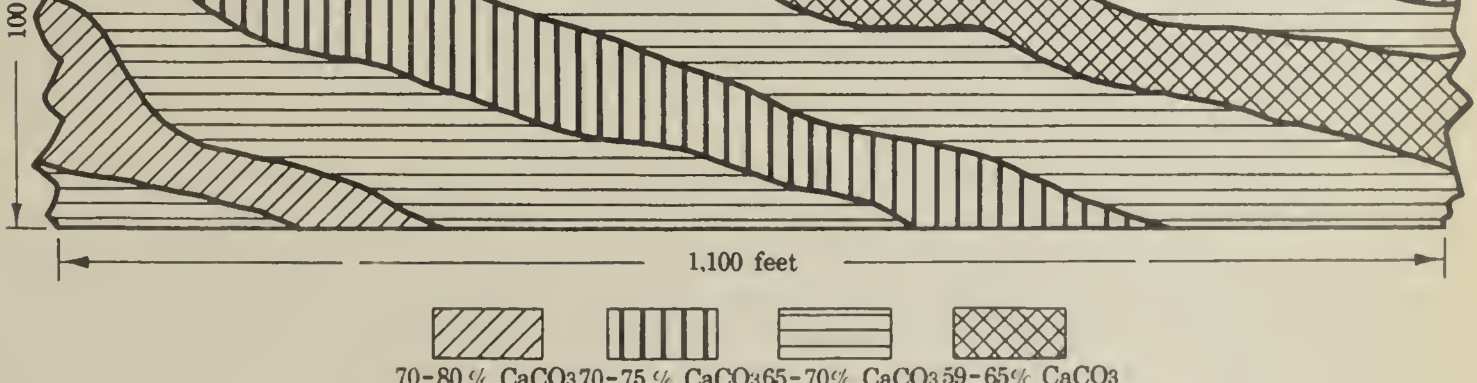


Figure 3. – Longitudinal section of rock below the quarry floor

vicinity of Egypt, and the limestone used at the Giant Portland Cement Co.'s plants is purchased and shipped from Annville, Pa. The cost delivered at the mill is $2.40 per long ton, or about seven times as much as the cost of the cement rock. Therefore it is not economical to work low-grade deposits of the latter.

RELIANCE QUARRY

In some places the cement rock outcrops, but as a rule it is covered by a clay overburden which varies in thickness from 1 to 20 feet. Sometines the clay extends to greater depth in pockets, chasms, fissures, or wide cracks in the otherwise solid rock. Occasional seams of quartz and soapstone are encountered but not in sufficient quantity or thickness to interfere with the method of quarrying.

Good cement rock is found to a depth of at least 230 feet within the area of the present workings. This area is located in the southeast section of the quarry (see fig. 1). The strata dip in a northwesterly direction at an angle of about 5°. Faults are rare and folding occurs only in a few places. The composition of the rock varies considerably in localized sections but the method of quarrying reduces this variation so that the rock delivered to the mill will average between 70 and 75 per cent calcium carbonate. A plan of the quarry and the track layout is shown in Figure 1. A section of the present face showing location of strata is shown in Figure 2. A longitudinal section of the rock below the floor of the quarry is shown in Figure 3.

PROSPECTING

Considerable prospecting has been done on the Reliance and adjacent properties of the company. One gas churn drill is used continually for this purpose. Test holes 6 inches in diameter are drilled to a depth of from 100 to 200 feet. In new properties the holes are first drilled on the corners of a 250-foot square and if the results of the tests indicate that further prospecting is necessary, the spacing is reduced, either for the purpose of tracing high-grade rock when it is encountered or to eliminate any doubts as to the value of the prospected area. Numerous prospect holes have been drilled from the surface of the Reliance quarry to a depth of 230 feet. The quarry floor has been prospected to a depth of 100 feet. As the present face is moved ahead, additional test holes are sunk in the new floor area on 100-foot squares, and the results are tabulated and recorded for future reference.

Loomis No. 4 clipper drill is the type used. The average footage drilled per day of 10 hours is approximately 40 feet. The speed varies with the strata encountered, which consists of clay, cement rock, veins of limestone and quartz, dolomite, and slate. The dolomite is the hardest rock encountered in large masses. The rate of drilling becomes slower with increasing depth. When water is encountered in the drill holes the footage drilled in unit time is much less than the average. The average cost of drilling prospect holes is $0.405 per foot. This cost includes labor (two men to a drill), material and fuel, repair labor, and also covers the time used in moving from one location to another.

METHODS OF SAMPLING AND ANALYSIS

Samples are taken from each 5 feet of drill hole. Each time the sludge is bailed out the first portion is thrown away and the second fill of the bailer is caught in a water pail; all other bailings are thrown away. The sample in the pail is washed free from clay,

is mixed well, and put into a small tin box or a tobacco can, several of which are provided for the purpose. Each can is marked with the number of the test hole and the depth at which the sample was taken.

Samples are dried and pulverized and a determination for carbonate, in terms of calcium carbonate, is made on each one by the simple and rapid acid-alkali method. After all samples from one hole are tested in the above manner a composite is made of all the samples of that hole. A carbonate determination is made on the composite; the result should agree with the arithmetical average of the individual results within 0.2 per cent. The actual lime in the composite, determined by the potassium permanganate method, is calculated to calcium carbonate. The difference (plus or minus value) between the actual calcium carbonate and the carbonate obtained by the acid-alkali method is added to the individual results so as to obtain the correct values. The composite samples are also analyzed for magnesia.

In using the acid-alkali method for the estimation of calcium carbonate in limestone, cement rock, and cement mixes, it is customary to standardize the acid and the alkali solutions with a standard sample of material approximating the composition of the samples to be tested. If the standard sample contains 42 per cent lime (75 per cent $CaCO_3$) and 2 per cent magnesia, a sample of rock which by this method analyzes say, 70 per cent calcium carbonate, contains the equivalent of 70 per cent $CaCO_3$ and 2 per cent MgO. But if the actual MgO is more than 2 per cent, then the $CaCO_3$ will be less than 70 per cent, and as the percentage of magnesia may vary in different sections and strata of a property, it is necessary to make a determination of the lime in the composite in order to check and make any correction of the acid-alkali determination.

When the acid-alkali result exceeds the actual calcium carbonate content, the difference, especially if considerable, can be attributed to a higher percentage of magnesia than that in the standard sample. This difference multiplied by 0.4 will equal approximately the increase in the percentage of magnesia. Other acid-soluble basic impurities affect the acid-alkali determination, but to a less degree.

ESTIMATION OF TONNAGE

Based on the analyses, areas are mapped out so that the average calcium-carbonate content of the whole area will exceed 70 per cent. The available tonnage is estimated on the basis of 155 pounds per cubic foot. Due to the fairly regular and solid formation of the rock, especially below the quarry floor, the tonnage can be estimated closely.

METHODS OF QUARRYING

The open-pit method of quarrying has been used from the beginning of operations in the Reliance quarry. The first cut was made at the mill site in order to excavate for the crusher and adjacent buildings. The area extending from the mill to the northwest was the first developed and worked due, no doubt, to the fact that it had less than 1 foot of overburden. When the good rock in this section became exhausted, or nearly so, the area to the southwest was prospected and developed. The overburden in this area consists of a yellow clay. The amount of clay per thousand tons of cement rock is approximately 100 cubic yards. This estimate applies only to the southwest area.

This heavy clay overburden is removed by stripping with a power shovel and a dragline scraper supplemented with some pick and shovel work. Due to the clay-filled crevices previously mentioned which, in part, can not be stripped by the regular methods with economy and safety, some waste is shot down with the rock. The waste removed in the quarry proper by the power shovels and transported to the No. 2 waste dump amounts to approximately 2 per cent of the rock recovered. A certain amount of waste is unavoidably mixed with the rock going to the mill but it is only harmful in that it requires a larger amount of the expensive limestone to regulate the mix.

Considering the nature of the overburden and its amount, the fact that 98 per cent of the material shot down can be recovered, that high and long working faces are possible, the open-pit method and the use of power shovels is logical and economical.

DRAINAGE

The present quarry floor is on the same level as the mill site but it is low with respect to the surrounding country. The maximum height of the quarry face above this level is 120 feet. However, due to the proximity and greater depth of other quarries, no drainage is necessary at present in the Reliance quarry. When the supply of cement rock above the present floor level is exhausted and if it is decided to sink deeper so as to recover the good rock below, it will then be necessary to provide for drainage.

STRIPPING

Stripping operations are carried on between April and October. The first cut is made with a Bucyrus steam shovel with a $\frac{3}{4}$-yard dipper. This is followed later by a Marion steam shovel which has been converted into a dragline scraper and equipped with a $\frac{3}{4}$-yard Page scraper bucket. Most of the clay is found in wide chasms and long crevices and the dragline is very suitable for removing it. In the narrow lateral crevices which are inaccessible to the scraper bucket several men are employed to dig the clay and shovel it into the path of the bucket. All overburden is removed as far as possible.

The clay is loaded directly into Easton side-dump cars. The cars hold $1\frac{1}{4}$ cubic yards and are hauled to the No. 1 waste dump by Vulcan steam locomotives. A train consists of six cars. The dump is located on the slope of a hill. The length of haul from the stripping area to the dump is between 1,500 and 2,000 feet; the grade is less than 2 per cent The track layout is shown in Figure 1. The gage is 30 inches and a 24-pound rail is used.

The average amount of overburden removed in 10 hours is 175 cubic yards; this figure applies to the dragline operation in which the stripping is tedious and expensive due to the awkward position of the clay. The dragline scraper and two locomotives are used at present. The stripping crew consists of 14 men as follows: 2 men on the dragline, 1 man on each locomotive, 6 pick-and-shovel men to supplement the scraper, and 4 men on the waste dump to empty cars and level off. One foreman has charge of stripping, drilling, and prospecting.

The total cost of stripping over a period of four years amounts to $0.599 per cubic yard.

Stripping operations are kept well ahead of drilling, and enough rock to supply the mill for about two years has been stripped.

MINING

In mining, the general plan is to develop long and high faces so as to bring down with one shot sufficient rock to supply the mill for several months. The Reliance mill requirements are between 25,000 and 30,000 long tons per month. The Central mill, when operating, requires about 18,000 tons per month. Both mills are supplied from the Reliance quarry. All the rock is used in the manufacture of cement. Waste inclusions amount to about 3 per cent of the rock. About two-thirds of this can be easily segregated, loaded separately, and transported to the No. 2 waste dump which is located close to the stripping dump but at a lower level on the side of the same hill. The haul to the lower waste dump is too steep to be made directly; it is accomplished by making one long level haul and three short hauls of 4 per cent grade. The total length of haul from the quarry face to the lower waste dump is approximately 3,500 feet.

The quarry face is perpendicular. From the east to the west side the height of the face increases from 85 to 110 feet. A circular face is developed. It is believed that this form of face gives the best blasting results with respect to the amount of explosives used per ton of rock for primary and secondary blasting.

PRIMARY DRILLING

Churn drills made by the Loomis Machine Co. are used for the primary drilling (and for prospecting). There are two gasoline drills, model JA, one traction and one half-caterpillar, and two steam drills, one traction and one horse-drawn. The gasoline drills are preferred as they are found to be more economical to operate and are more convenient with respect to supplies, such as fuel and water. A horse and cart delivers the coal from the mill to the steam drills, dragline, and to the locomotives. The driller's helper brings the gasoline in 5-gallon cans from the mill. A pipe line from the mill to the top of the quarry delivers either water or air. The air pressure is 65 pounds. The air is used for operating an I. R. jackhammer when it is found necessary to blast some of the rock in order to level the surface or to fill in some of the crevices prior to locating the churn drill. Where the crevices are wide, the churn drill rests on a cribbing made from railroad ties.

Oil-well drill steel $5\frac{5}{8}$ inches in diameter is used. Drill bits weigh 250 pounds and are dressed to $6\frac{1}{8}$ inches diameter. Drill stems measure $4\frac{1}{4}$ inches by 20 feet. The bits are dressed in an Armstrong bit dresser. Casing is used only in clay or loose material.

The average drilling speed is 45 feet per day of 10 hours. When in hard rock, bits have to be changed and dressed oftener so as to maintain a uniform diameter of the hole.

Holes are spaced 15 feet apart, parallel to the face, and cary a 28-foot burden. Where it is impractical, due to extreme irregularities of the top surface, to drill parallel to the face the holes approach or recede from it, depending on the apparent degree of solidity of the rock at that point. Formerly it was the practice to increase the spacing in the more solid rock but this procedure was discontinued as it was found to make the cost of secondary blasting too high. The number of holes drilled for one shot ranges from 35 to 45. The diameter of the holes is $6\frac{1}{4}$ inches; the depth is governed by the height of the face which varies from 85 to 110 feet. All holes are drilled 6 feet below the quarry floor.

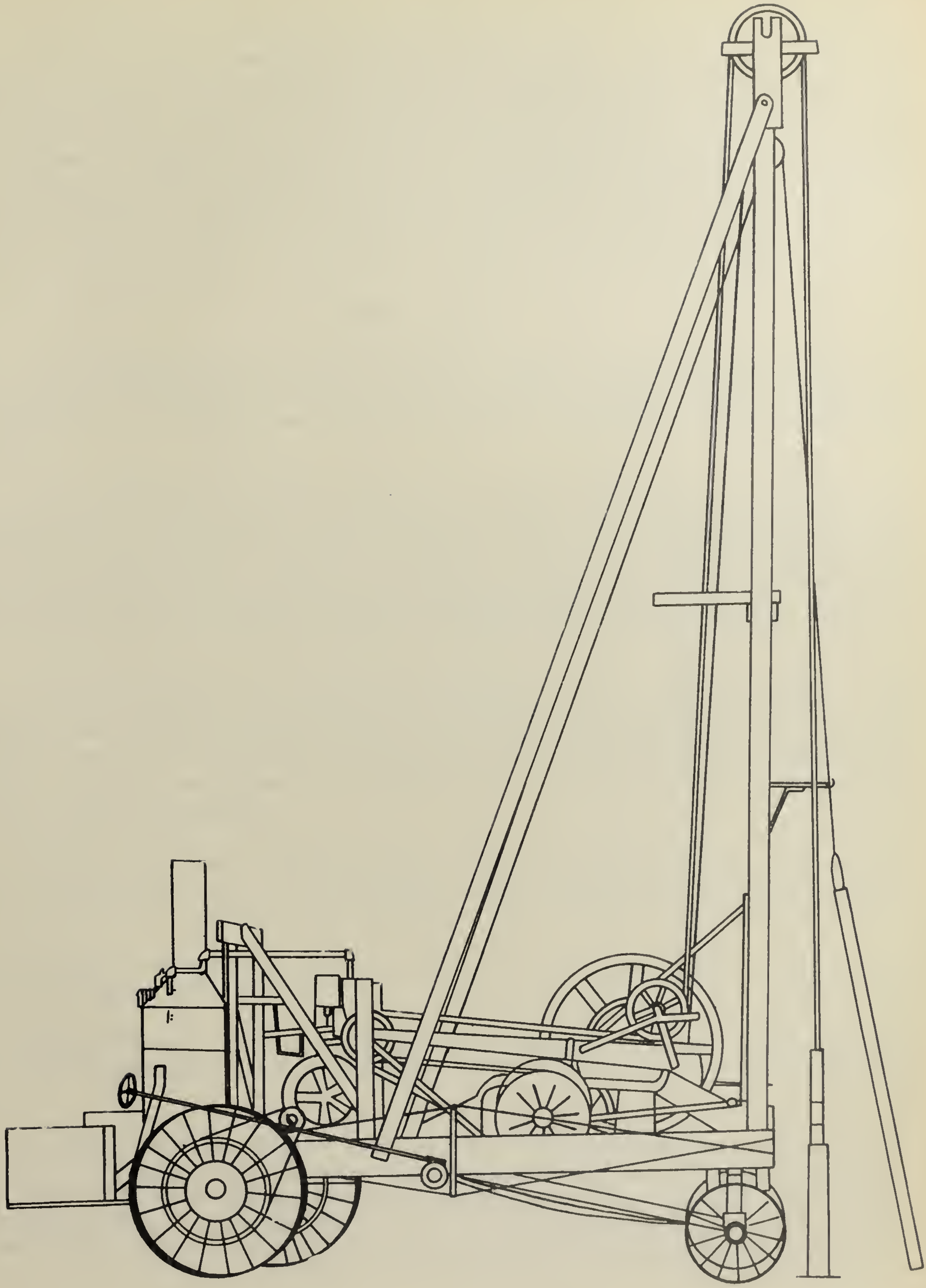


Figure 4.- Steam well drill

Drilling starts immediately after each shot is made. Positions of the holes are located by measuring from permanent stakes the exact distances toward the quarry face; this establishes the base line for the row of holes. A record is kept of the location of each set of holes and the position of each hole is plotted on a blue print of the quarry. Each hole is numbered. Samples are taken every 5 feet and are marked according to the hole number and the depth at which the samples were taken. The samples are treated as described under "Methods of Analysis." They are stored until after the shot is made and the stone used. A composite is made of samples of all holes in the blast, and a complete analysis is made of it.

PRIMARY BLASTING

The explosives used for primary blasting are 60 and 40 per cent gelatine dynamites. The cartridges are 5 inches in diameter and 24 inches long. Cordeau is used for detonating the charge. No. 6 detonators and Beaver fuse (speed 40 seconds per foot) are used for firing.

The powder used for primary blasting is not kept in storage but is delivered to the quarry by auto trucks on the day the shot is to be fired. It is distributed to all the holes in proportion to the amounts and the strengths required. The boxes are opened by extra labor (not the loading crew). There are from two to four loading crews of three men each. They fill the holes in rotation and under the the supervision of the explosive company's representative who acts as powderman and who is responsible for the shot.

In loading each hole the cordeau is attached to the bottom stick of powder. As a rule the bottom is loaded with the 60 per cent powder and the top with 40 per cent. The thickness of the burden at different heights also governs the distribution of the different strengths of explosive. When pockets are encountered, that portion of the hole is filled with clay. The holes are loaded to within 20 to 25 feet from the top and the remainder is filled with clay and tamped down.

After all holes are loaded the ground is cleared of powder boxes and other obstructions, a trunk line of cordeau is run from the first to the last hole, and the free end of the line is connected to the detonator and fuse. The fuse is lighted with a match.

All holes are loaded and fired on the same day. Between 10 and 15 minutes are required to load each hole.

The fragmentation desired is dependent on the size of the primary crusher which is a 48 by 60 inch jaw type. There is no minimum limit as all the stone is used for making cement.

In the drilling and blasting operations holes have been drilled with 16-foot centers and 24 feet of burden; 18-foot centers and 26 feet of burden; and different strengths of explosives, 30, 40, 50, and 60 per cent have been used under these varied conditions. The most economical results with respect to the total cost of primary and secondary blasting have been obtained by drilling the holes 15-foot centers and carrying a 28-foot burden.

SECONDARY DRILLING AND BLASTING

Ingersol-Rand jackhammers, type BCR4, are used for drilling the large bowlders. The air is delivered from the compressor to the quarry over a 2-inch line. Several $1\frac{1}{4}$-inch air lines branch out from the main line to different sections, and $\frac{3}{4}$-inch leads connect the main line to the drills. The air pressure at the drills is between 70 and 80 pounds.

The drill steel is $\frac{7}{8}$-inch hexagon stock; $1\frac{1}{4}$-inch bits are used for $1\frac{1}{2}$-inch diameter holes, and $1\frac{3}{4}$-inch bits for 2-inch holes. In large bowlders in which the depth of the holes exceeds 10 feet, the holes are made 2 inches in diameter; shallower holes are made $1\frac{1}{2}$ inches in diameter. The drilling speed is about 1 foot in 4 minutes.

The large bowlders encountered by the steam shovels are cast to one side and drilled by a crew of two men used for this purpose and for barring down loose bowlders which are also drilled. Holes are drilled to the centers of the bowlders. The dynamite used is $1\frac{1}{4}$ inch diameter 40 per cent gelatin. No. 6 detonators and Beaver fuse (speed 40 seconds per foot) are used for firing. The holes are filled to within 2 inches of the top with the powder, the cap and fuse are attached to the last stick, and the last 2 inches of the hole is filled with clay and tamped well. The fuses are cut to not less than 30 inches and are lighted with a wick.

In primary blasting the shots will average 3.7 tons per pound of explosive; in secondary blasting the ratio is approximately 1 pound of explosive to 44 tons of stone.

LOADING STONE

Two Bucyrus No. 70 traction-type steam shovels are used for loading the stone. The dippers are $2\frac{1}{2}$-yards capacity. One Bucyrus No. 65 is held in reserve. Two shovels are able to load between 180 and 190 long tons per hour; this includes the intermittent stops caused by transportation delays. There are 3 men on each shovel; an engineer, a fireman, and a craneman. Working 8 hours per day and 5 days per week, the mill requirements of from 25,000 to 30,000 long tons per month can be supplied.

TRANSPORTATION

Three Vulcan steam locomotives running on standard gage 60-pound rail are used to transport the rock to the crusher. The grade to the crusher is practically level. Atlas 10-ton side-dump cars are used. The load in each car is $7\frac{1}{2}$ long tons. Five cars make up a trainload. The fuel used on the locomotives, shovels, etc., is run-of-mine coal. It is hauled by a horse and cart to the shovels from a storage pile located on the track adjacent to the pump house. The locomotives obtain their coal from the same source. The water for the boilers is treated by the lime-soda process.

From the crusher to the quarry face the locomotives follow the horseshoe track around the face and make the return trip with the load over the straight piece of track (ST, fig. 1). The cars receive half of their load from the shovel nearer to the mill, and the load is completed by the further shovel. This loading system helps to mix the rock and reduces the variation in the composition. The haul to the crusher is about 1,200 feet.

The total quarry crew (exclusive of the men on primary drilling, blasting, and stripping) consists of 17 men; 3 on each shovel, 1 man on each locomotive, 2 powdermen for

secondary drilling and blasting and for barring down, 2 pitmen, 2 track repairmen, 1 man at the crusher to dump cars, and 1 foreman.

CRUSHING PLANT

The primary crusher is located below the level of the tracks and rests on a solid rock foundation. Figure 5 is a sectional view of the silos and the crushing equipment.

A small drum hoist geared to a 15-hp. motor is used for unloading the cars. The lift is applied through an overhead block and tackle. The cars are dumped directly on to a 5 by 12 foot Traylor Sheridan grizzly feeder which is driven by a variable-speed 25-hp. motor. The feeder has a reciprocating motion and carries the rock ahead slowly and uniformly to a 48 by 60 inch Traylor jaw crusher belted to a 200-hp. motor. The fines, 3 inches and under, pass through the grizzly bars. The by-passing of the fines which contain most of the clayey material prevents choking of the crusher. The capacity of the primary crusher is 250 tons per hour crushing to 8 inches.

The crushed product and the fines which pass the grizzly are picked up by a 48-inch by 60-foot link-belt bucket elevator and delivered to the secondary crusher, a 42 by 48 inch Jeffrey swing-hammer mill directly connected to a 200-hp. motor. No intervening storage bin is required. The Jeffrey mill is very efficient for crushing cement rock. In order to reduce dust losses in the dryer stack several of the mill hammers have been removed.

The Jeffrey product, 3 inches and under in size, drops through a side chute to the lower run of a link-belt carrier which is driven by a 50-hp. motor. The buckets are 30 by 36 inches and overlap each other. The capacity of the carrier is 300 tons per hour. The carrier passes over a Merrick weightometer which weighs and records the quantity of rock going to the storage silos.

There are eight round concrete silos (two rows of four) with an intersticial row star shaped. The storage capacity is 12,000 tons. One section of the silos is 65 feet deep (inside depth) and the other is 40 feet deep. The latter is built over the railroad tracks, and double duplex gates 24 by 24 inches are attached to the bottom of these four silos. This permits the crushed rock to be loaded into gondolas and shipped to the Central mill of the company.

A traveling tripper is used for dumping the carrier buckets containing the crushed rock into any of the silos. A small back-geared drum hoist driven by a 2-hp. motor pulls the tripper in a counter direction to that of the carrier; the forward movement is effected by the pull of the buckets.

The empty buckets descend at the rear end of the silos and travel through a tunnel under the latter. Under each silo there is a link-belt, reciprocating feeder. The feeders are driven through chain drives and clutches from a line shaft which is driven by a variable-speed 10-hp. motor. They can be adjusted to withdraw the rock at a uniform rate from any number of silos at the same time. By feeding from several silos simultaneously, the rock is blended and the combined flow of material is more uniform in composition than that which entered the silos.

The feeders deliver the rock to the lower run of the carrier which conveys it to a point ahead of the secondary-crusher discharge chute. At this point a tripper dumps the

blended rock into the boot or pit of a 36-inch by 96-foot link-belt bucket elevator (elevator No. 2, fig. 5) driven by a 40-hp. motor. The empty carrier buckets travel a few feet forward and pick up the product of the secondary crusher as already described. In case of a shut-down of the carrier the product of the secondary crusher or the crushed rock in the two north end silos can be fed directly to the No. 2 elevator which delivers the rock to the mixer bins.

In the crushing department there are five men and a foreman. The three silo men, one on each 8-hour shift, keep the mixer bins supplied with cement rock and limestone; the latter is stored in two of the silos. One man operates the upper tripper (which distributes the rock to the silos) and collects an average daily sample of the rock from the carrier buckets. The sixth man helps in the unloading of cars of limestone and does odd jobs.

DRYING AND MIXING

There are two reinforced concrete mixer bins each having a capacity of 200 tons. One is used for cement rock and the other for the crushed limestone. The mixing room is located under thses bins. Here the cement rock and the limestone are proportioned to make the raw cement mixture. The bins are equipped with duplex gates through which the stone feeds into the hopper of the Toledo scale. The hopper is drop-bottom and discharges at a uniform rate onto a horizontal belt conveyor, 39 inches by 7 feet 3 inches, which carries the mixture to a 16-inch by 35-foot overlapping bucket elevator. The latter delivers the product onto another horizontal belt conveyor 36 inches by 14 feet, to be discharged into the cylindrical dryer. The head pulley of the latter conveyor is a 16-inch diameter magnetic separator which picks the tramp iron from the stone.

The dryer, 8 by 80 feet, is direct fired by pulverized coal and can dry 67 tons per hour. Between 1,100 and 1,300 tons are dried each 24 hours. The dryer, the bucket elevator, the belt conveyors, and the auxiliary conveying equipment for cleaning the dryer stack chamber, are driven by a 75-hp. motor through a Cleveland Worm and Gear Co. speed reducer.

The dryer discharges directly into a 42 by 48 inch Jeffrey hammer mill direct-connected to a 75-hp. motor. Originally this mill was one of two installed for secondary crushing. One was found to be sufficient and the other was used to replace a smaller type of hammer mill at the discharge end of the dryer. The main purpose of this tertiary crushing is to break any stray large lumps of stone. The rock is reduced to under 1-inch size and discharges into an inclosed 17-inch by 40-foot bucket elevator which elevates it to a large rectangular steel bin located over the preliminary grinding mills. The bin has a capacity of about 100 tons.

There are three men in the mixing department, one man on each shift of 8 hours. There are no men in the drying department. The dryer, once started, requires little attention and that is given by the men in the grinding department.

GRINDING

The preliminary grinders are two Bradley Hercules mills. Each is direct-connected to a 300-hp. synchronous motor. The mills are equipped with 9-mesh screens and have an output of 40 long tons per hour per mill.

Screen Analysis of Hercules Product

Mesh	Per cent undersize
200	54.2
100	62.4
80	68.1
50	75.0
30	88.2
20	98.5
10	100.0

The ground material discharges into a 24-inch by 45-foot bucket elevator. The latter discharges into a 24-inch by 60-foot screw conveyor which distributes the material into a steel blending bin. The elevator and conveyor are chain driven from a Palmer-Bee Co. speed reducer coupled to a 25-hp. motor. There are five discharge openings in the conveyor casing. The first four are equipped with steel slides which are operated by an electrical device that opens and closes them in rotation. This insures a uniform distribution of the material entering the blending bin. The lower section of the bin consists of five hoppers, the centers of which are directly under the conveyor openings. Under the hoppers there are five rotary valves driven through chain drives from a line shaft. The valves withdraw the mix from the bin at a uniform rate and the five streams feed into an 18-inch by 50-foot screw conveyor. This discharges into a 17-inch by 25-foot bucket elevator which in turn discharges into an 18-inch by 66-foot screw conveyor which feeds the tube mills. The valves, the two 18-inch screw conveyors, and the last elevator are all driven by a 25-hp. motor through a Palmer-Bee Co. speed reducer.

There are two tube mills for the final grinding of the raw mix. One is a 7 by 26-foot Traylor and the other is a No. 20 7-foot by 23-foot 6-inch Smidth. The former is driven by a 500-hp. synchronous motor; the latter by a 400-hp. synchronous motor. The mills are charged with $\frac{3}{4}$-inch steel balls and cylpebs. The charge in the large mill is 35 tons; in the smaller mill 24 tons. The combined output of both mills is about 50 tons per hour. Ninety per cent of the product passes a 200-mesh screen.

The grinding department operates 24 hours per day, seven days per week. Six men, two on each 8-hour shift, are employed in this department.

REPAIR SHOP

There is a large and well-equipped repair shop at one end of which is the blacksmith's shop. A 12 by 10 inch, class 38, Pennsylvania Pump and Compressor Co. compressor supplies the air for the jackhammers and for miscellaneous purposes. An Armstrong bit dresser for the well-drill bits and a Denver Rock and Drill Co. bit dresser for the air-drill bits are part of the blacksmith shop equipment.

Repairs on the shovels and the locomotives are made without delay. The men operating these units work on a bonus system and the equipment for mining and transportation is kept in good condition at all times. All boilers are cleaned out every two weeks.

POWER

Electrical power is purchased from the Pennsylvania Power and Light Co. The substation equipment consists of two 3,000 kva. transformers and two 100-kw., 220-volt, direct-current motor generators. The incoming power is stepped down from 66,000 to 440 volts.

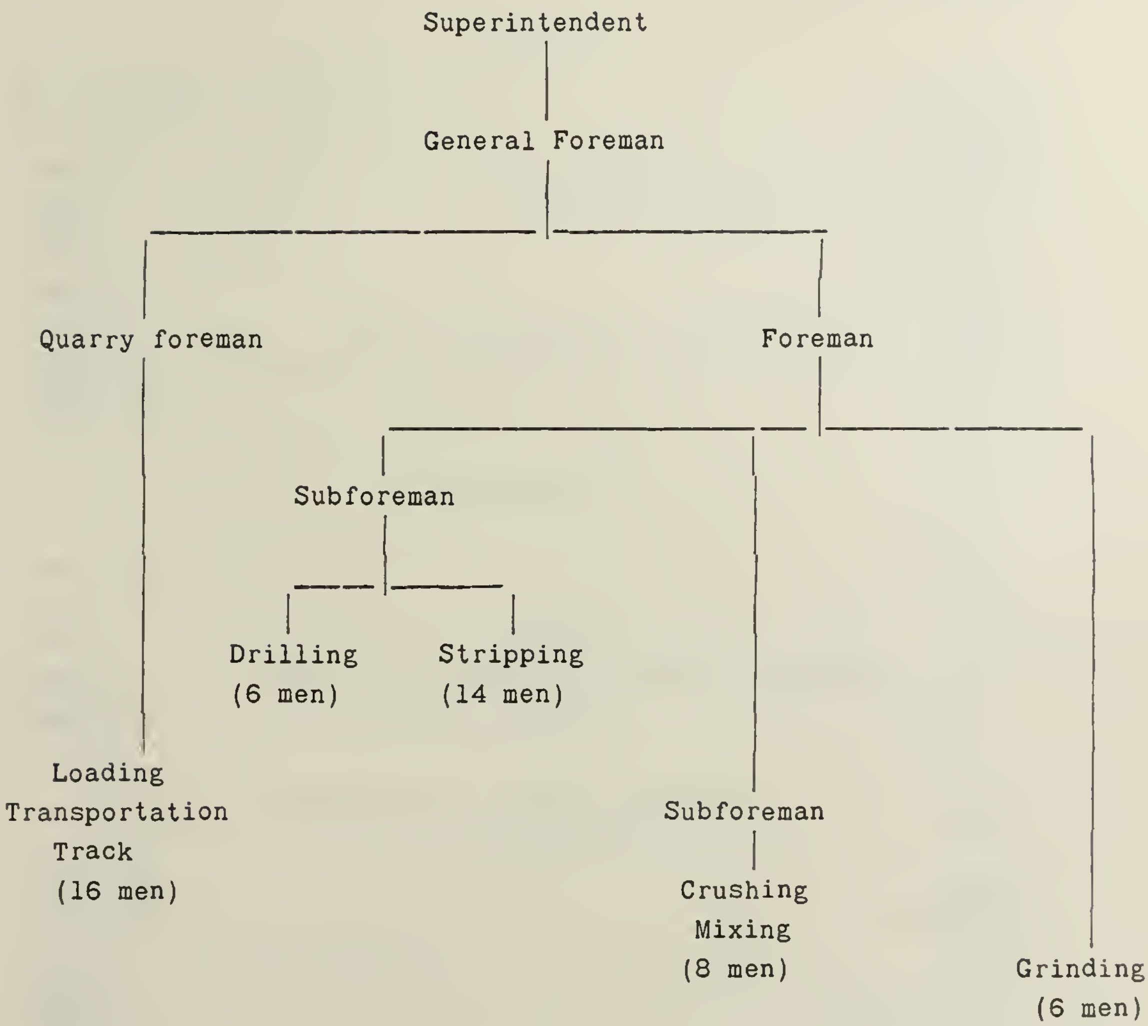
PAY SYSTEM

The men employed on stripping, primary drilling and blasting, and in the crushing and grinding departments are paid a straight hourly wage. The crew engaged in transportation, loading, and unloading of the cement rock is paid an hourly wage and a bonus. There are 16 men included in the bonus distribution. The bonus is based on a fixed labor cost and the actual labor cost for transportation, loading and unloading. A maximum labor cost of 8 cents per long ton is the present fixed standard. The difference between the maximum and the actual labor cost is paid to the men.

The wage scale is as follows:

Position	Number	Rate per hour
Shovel engineers	3	$0.53
Shovel firemen	3	.47
Shovel cranemen	3	.48
Well drillers	3	.495
Well drillers' helpers	3	.425
Drillers (jackhammer)	2	.505
Pitmen	2	.45
Locomotive engineers	5	.51
Trackmen	2	.465
Men on waste dump	4	.40
Pick and shovel men on stripping	6	.40
Blacksmith	1	.60
Tripper man (mill)	1	.45
Silo men	3	.46
Mixer men	3	.425
Millers	3	.58
Millers' helpers	3	.53
Subforemen	2	.58

ORGANIZATION CHART



COSTS

	Period covered, 1929.
Production of stone (short tons)	288,528
Overburden removed (cubic yards)	26,956

1. Summary

	Cost per ton
Stripping	$0.05599
Primary drilling and blasting	.06606
Secondary drilling and blasting	.00855
Loading	.07454
Transportation	.05258
Primary and secondary crushing and storage	.07720
Drying, mixing, tertiary crushing and grinding	.36378
Total	$0.69870

2. Stripping

Labor	$0.03627
Purchased power	.00004
Fuel	.00467
Other supplies	.01500
Total	$0.05598

3. Primary drilling and blasting

Labor	$0.01015
Explosives	.04571
Fuel	.00206
Other supplies	.00814
Total	$0.06606

4. Secondary drilling and blasting

Labor	$0.00600
Explosives	.00250
Other supplies	.00005
Total	$0.00855

5. Loading stone (shovels)

Labor:	
Operating	$0.02200
Repair	.00450
Pitmen	.00560
Cleaning quarry	.00088
Teaming	.00375
Foreman	.01082
Purchased power	.00030
Fuel	.00973
Other supplies	.01696
Total	$0.07454

6. Transportation

	Cost per ton
Labor:	
Operating locomotives	$0.02140
Repairing locomotives	.00481
Repairing tracks	.00239
Moving tracks	.00965
Repair material:	
Locomotives	.00346
Tracks	.00155
Moving tracks	.00032
Purchased power	.00030
Fuel	.00766
Oil and waste	.00104
Total	$0.05258

7. Primary and secondary crushing and storage

Labor:	
Operating	$0.04168
Repairs	.00293
Supplies:	
Operating	.00033
Repairs and maintenance	.01045
Oil and waste, tools	.00745
Fuel	.00063
Purchased power	.01373
Total	$0.07720

8. Drying, mixing, tertiary crushing and grinding

Labor:	
Operating	$0.05449
Repair	.03649
Supplies:	
Operating	.00062
Repairs and maintenance	.03158
Tools	.00007
Oil and waste	.00753
Fuel	.02947
Purchased power	.20353
Total	$0.36378

9. Summary of cost distribution

	Labor	Power	Fuel	Explosives	Other supplies	Total
Stripping	$0.03627	$0.00004	$0.00467		$0.01500	$0.05598
Primary drilling and blasting	.01015		.00206	$0.04571	.00814	.06606
Secondary drilling and blasting	.00600			.00250	.00005	.00855
Loading stone	.04755	.00030	.00973		.01696	.07454
Transportation	.03825	.00030	.00766		.00637	.05258
Primary crushing		.00574				
Secondary crushing		.00348				
Storage in		.00180				
Storage out	.04461	.00271	.00063		.01823	.07720
Mixing, drying		.01864				
Tertiary crushing		.00401				
Preliminary grinding		.04963				
Final grinding		.12824				
Conveying, etc.	.09098	.00301	.02947		.03980	.36378
Totals	$0.27381	$0.21790	$0.05422	$0.04821	$0.10455	$0.69869

10. Man-hours

	Man-hours per ton	Tons per man-hour
Stripping and drilling	0.13900	7.19
Loading and transportation	.12608	7.93
Crushing	.11696	8.55
Drying and grinding	.17914	5.58

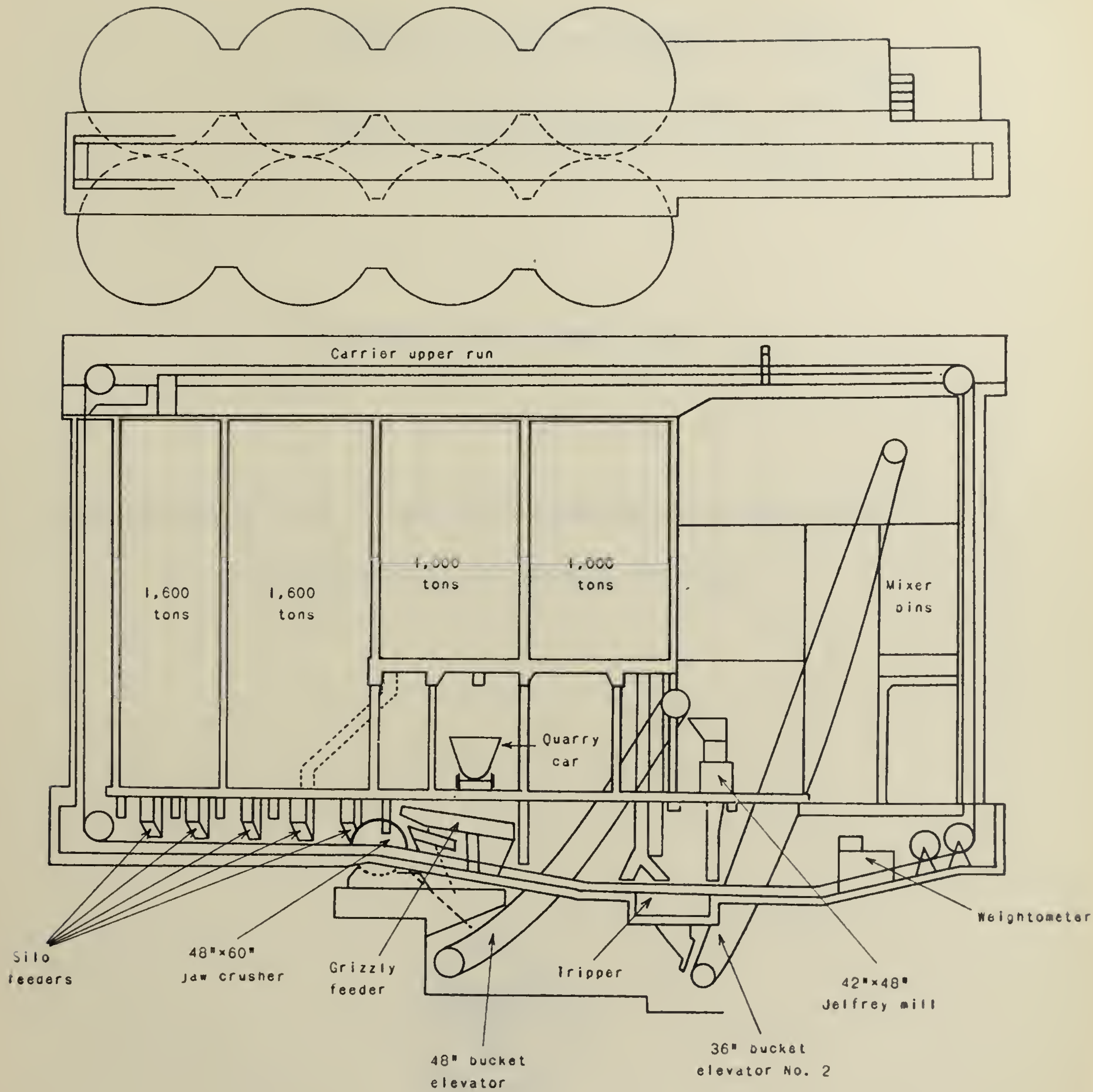


Figure 5.- Sectional view of silos and crushing plant

I. C. 6449

April 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

BIBLIOGRAPHY OF THE METALLURGICAL WORK OF THE U. S. BUREAU OF MINES IN 1930

BY

R. S. Dean

I.C.6449.
April, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

A BIBLIOGRAPHY OF THE METALLURGICAL WORK OF THE U. S. BUREAU OF MINES IN 1930[1]

By R. S. Dean[2]

General Papers:

A general survey of the metallurgical work of the U. S. Bureau of Mines has been given in an article in the Missouri School of Mines Alumnus, and a somewhat more detailed review of the bureau work on lead and zinc will be found in the Mining Congress Journal. C. C. Furnas has contributed one general mathematical article.

Dean, R. S. Program of Metallurgical Investigations of the U. S. Bureau of Mines. Missouri School of Mines Alumnus, vol. 4, No. 4, 1930, pp. 7-8.

__________ Investigations of the U. S. Bureau of Mines on the Milling and Smelting of Lead and Zinc. Min. Cong. Jour., vol. 16, 1930, p. 799, 808.

Furnas, C. C. Evaluation of the Modified Bessel Function of the First Kind and Zeroth Order. Am. Mathematical Monthly, vol. 37, No. 6, 1930, pp. 282-287.

Fundamental Studies:

The systematic study of thermodynamic properties of metallurgically important substances has continued at the Pacific Experiment Station, Berkeley, Calif., and papers have appeared giving specific heat data on manganese sulphide, ferrous sulphide, calcium sulphide, silicon, arsenic, arsenic trioxide, arsenic pentoxide, antimony, antimony trioxide, antimony pentoxide, bismuth, and bismuth trioxide. A critical survey of the specific heat of water vapor has been published and a general paper on the use of these data to study metallurgical reactions.

Anderson, C. T. The Heat Capacities of Arsenic, Arsenic Trioxide and Arsenic Pentoxide at Low Temperatures. Jour. Am. Chem. Soc., vol. 52, 1930, pp. 2296-2300.

______________ The Heat Capacity of Silicon at Low Temperatures. Jour. Amer. Chem. Soc., vol. 52, 1930, pp. 2301-2304.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6449."

2 - Chief engineer, metallurgical division, U. S. Bureau of Mines.

Anderson, C. T. The Heat Capacities at Low Temperatures of Antimony, Antimony Trioxide, Antimony Tetroxide and Antimony Pentoxide. Jour. Am. Chem. Soc., vol. 52, 1930, pp. 2712-2720.

______________ The Heat Capacities of Bismuth and Bismuth Trioxide at Low Temperatures. Jour. Am. Chem. Soc., vol. 52, 1930, pp. 2720-23.

Dean, R. S. Use of Thermodynamical Data to Study the Chemical Reactions of Metallurgical Processes. Inf. Cir. 6395, Bureau of Mines, 1930, 12 pp.

Eastman, E. D. Specific Heat of Water Vapor at High Temperatures Derived from Explosion Experiments. Inf. Cir. 6337, Bureau of Mines, 1930, 16 pp.

Hincke, W. B. The Vapor Pressure of Antimony Trioxide. Jour Am. Chem. Soc., vol. 52, 1930, pp. 3869-77.

Maier, C. G. The Heat of Formation of Zinc Oxide. Jour. Am. Chem. Soc., vol. 52, 1930, pp. 2159-2170.

Ore Dressing:

The bureau has continued its investigations of grinding and classification, and three papers have appeared by Fahrenwald. One more paper by Gaudin on flotation fundamentals has appeared. A comprehensive investigation of iron ore beneficiation has been started, and papers from the Mississippi Valley Experiment Station, Rolla, Mo., have appeared covering investigations of Missouri iron ores and ores of the Cuyuna district.

Clemmer, J. B., and Coghill, W. H. An Improved Laboratory Elutriator and Its Application to Ores. Eng. and Min. Jour., vol. 129, 1930, pp. 551-554.

Coghill, Will H. The Classification and Tabling of Difficult Ores, with Particular Attention to Fluorspar. Tech. Paper 456, Bureau of Mines, 1930, 40 pp. 15 cents.

Coghill, Will H., and Anderson, C. O. Notes on Free Gangue in Tri-State Ores. Joplin (Mo.) Globe, Annual Mining Number, February 2, 1930, p. 41.

DeVaney, F. D., and Cooke, S. R. B. Laboratory Concentration of the Missouri Iron Ores of Iron Mountain and Pilot Knob. Cooperative paper of the U. S. Bureau of Mines and the Missouri School of Mines and Metallurgy, University of Missouri, Bull., vol. 11, No. 3, 1930, 38 pp.

DeVaney, F. D., and Clemmer, J. B. Concentration Tests on the Manganiferous Iron Ores of the Cuyuna District, Minnesota. Rept. of Investigations 3045, Bureau of Mines, 1930, 9 pp.

DeVaney, F. D., and Coghill, W. H. Concentration Tests on Tailings from the Washing Plants of the Mesabi Range, Minnesota. Rept. of Investigations 3052, Bureau of Mines, 1930, 23 pp.

Fahrenwald, A. W. Grinding and Classification.- I. Batch Grinding. Rept. of Investigations 2989, Bureau of Mines, 1930, 9 pp.

________________ Grinding and Classification.- II. Batch Closed Circuit Grinding. Rept. of Investigations 2990, Bureau of Mines 1930, 11 pp.

Fahrenwald, A. W., and Staley, W. W. The Power Consumed by Rotating Disks and Other Shaped Objects in Fluid Mediums. Rept. of Investigations 3006, Bureau of Mines, 1930, 7 pp.

Gaudin, A. M., Haynes, C. B., and Haas, E. C. Flotation Fundamentals (IV). Coop. Tech. Paper 7, U. S. Bureau of Mines and University of Utah, 1930, 40 pp.

Gaudin, A. M., and Anderson, A. E. Flotation Fundamentals (V). Coop. Tech. Paper, U. S. Bureau of Mines and University of Utah, 9, 1930, 25 pp.

<u>Iron and Steel</u>:

The investigation on the physical chemistry of steel making has proceeded with very interesting results, which have been published in a series of papers. Blast furnace operation has also been studied, and papers published covering resistance of ores to decrepitation and studies of gas flow and heat transfer in the blast furnace. The present status of the sponge-iron process has been summarized by Barrett.

Barrett, E. P. Sponge Iron and Its Relation to the Steel Industry. Min. and Met., vol. 11, 1930, pp. 395-396. Abstract entitled "Sponge Iron Process Not a Competitor of the Blast Furnace," Iron Age, vol. 126, No. 26, 1930, p. 1901.

Furnas, C. C. Heat Transfer from a Gas Stream to a Bed of Broken Solids (II). Ind. and Eng. Chem., vol. 22, 1930, pp. 721-731.

Herty, C. H., jr. Fundamental and Applied Research on the Physical Chemistry of Steel Making. Rept. of Investigations 3054, Bureau of Mines, December, 1930, 12 pp. Metals and Alloys, vol. 1, 1930, pp. 883-889.

________________ Iron and Steel Metallurgy. Annual Survey of American Chemistry, vol. 4, 1930, pp. 160-165.

Herty, C. H., jr., Christopher, C. F., and Stewart, R. W. The Physical Chemistry of Steel Making: Deoxidation with Silicon in the Basic Open-Hearth Process. Coop. Bull. 38, U. S. Bureau of Mines, the Mining and Metallurgical Advisory Boards, and the Carnegie Institute of Technology, 1930, 172 pp.

Herty, C. H., jr., Fitterer, G. R., and Byrns, J. M. The Physical Chemistry of Steel Making: Deoxidation of Steel with Aluminum. Coop. Bull. 46, U. S. Bureau of Mines, the Mining and Metallurgical Advisory Boards, and the Carnegie Institute of Technology, 1930, 45 pp.

Herty, C. H., jr., Gaines, J. M., Freeman, H. and Lightner, M. W. A New Method for Determining Iron Oxide in Liquid Steel. Trans. Am. Inst. Min. and Met. Engrs., Iron and Steel Division, 1930, pp. 28-38; Am. Inst. Min. and Met. Engrs. Tech. Pub. 311, 1930, 13 pp.; Abstract in Blast Furnace and Steel Plant, vol. 18, 1930, pp. 468-71.

Herty, C. H., jr., Hartgen, F. A., Heidish, J. A., Metcalf, K., Norris, F. G., and Royer, M. B. Temperature-Viscosity Relations in the Lime-Silica System. Coop. Bull. 47, U. S. Bureau of Mines, the Mining and Metallurgical Advisory Boards, and the Carnegie Institute of Technology, 1930, 27 pp.

Joseph, T. L., and Barrett, E. P. The Resistance of Iron Ores to Decrepitation and Mechanical Work. Am. Inst. Min. and Met. Eng., Tech. Pub. 372, 1930, 15 pp.

Hydrometallurgy:

The work on leaching copper ores has been continued. The outstanding development is the leaching of slimes by agglomeration.

Keyes, H. E. Innovations in Copper Leaching Employing Ferric Sulphate-Sulphuric Acid. Bull. 321, Bureau of Mines, 1930, 67 pp., 20 cents.

Sullivan, J. D. The Chemistry of Leaching Chalcocite. Tech. Paper 473, Bureau of Mines, 1930, 24 pp. 10 cents.

_______________ The Chemistry of Leaching Bornite. Tech. Paper 486, Bureau of Mines, 1930, 20 pp.

_______________ The Chemistry of Leaching Covellite. Tech. Paper 487, Bureau of Mines, 1930, 18 pp. 5 cents.

_______________ Apparatus for Circulating Liquids. Eng. and Min. Jour., vol. 130, 1930, p. 389.

Sullivan, J. D., Oldright, G. L., and Keck, W. E. Method for Measuring Voids in Porous Materials. Rept. of Investigations 3047, Bureau of Mines, 1930, 8 pp.

Sullivan, J. D., and Towne, A. P. Agglomeration and Leaching of Slimes and Other Finely-Divided Ores. Bull. 329, Bureau of Mines, 1930, 60 pp., 15 cents.

Manganese Investigations:

Investigation of both pyrometallurgical and hydrometallurgical methods of recovering manganese from domestic ores has been continued and progress reports have been published covering certain phases of both processes.

Davis, C. W. Dissolution of Various Manganese Minerals. Rept. of Investigations 3024, Bureau of Mines, 1930, 11 pp.

____________ The Action of Sulphur Dioxide on Manganese Oxides at Elevated Temperatures. Rept. of Investigations 3033, Bureau of Mines, 1930, 16 pp.

Healey, M. V., and Johns, A. L. Selected Bibliography and Map of Manganese Deposits of the United States by Districts. Inf. Cir. 6274, Bureau of Mines, 1930, 19 pp.

Herty, C. H., jr., Conley, J. E., and Royer, M. B. Study of High-Manganese Slags in Relation to the Treatment of Low-Grade Manganiferous Ores. Rept. of Investigations 3048, Bureau of Mines, 1930, 4 pp

Joseph, T. L., Barrett, E. P., and Wood, C. E. Experiments Demonstrate Method of Producing Artificial Manganese Ores. Am. Inst. Min. and Met. Engrs., Tech. Pub. 310, 1930, 29 pp. Extract in Iron Age, vol. 125, 1930, pp. 723-724; Blast Furnace and Steel Plant, vol. 18, 1930, pp. 631-7; Abstracted in Iron and Coal Trades Rev., vol. 120, No. 3243, 1930, pp. 686-687.

Rare and Precious Metals Investigations:

The bureau has been very active in the department of rare and precious metals, and work on a wide variety of subjects has been reported. The report on processes of extracting radium from carnotite prepared for the House Committee on Mines and Mining is to be particularly mentioned.

Doerner, H. A. Processes for Extracting Radium from Carnotite. Rept. of Investigations 3057, Bureau of Mines, 1930, 35 pp.

____________ Concentration of Chromite. Rept. of Investigations 3049, Bureau of Mines, 1930, 8 pp.

____________ The Roasting of Chromite Ores to Produce Chromates. Rept. of Investigations 2999, Bureau of Mines, 1930, 30 pp.

____________ Notes on the Determination of Molybdenum. Inf. Cir. 6335, Bureau of Mines, 1930, 3 pp.

____________ Possibilities of Production of Radium and Vanadium from Carnotite. Ind. and Eng. Chem., vol. 22, 1930, pp. 185-189.

Keck, W. E., Oldright, G. L., and Shelton, F. K. A Method for the Determination of Cadmium in Mill and Smelter Products. Coop. Tech. Paper 12, Bureau of Mines and University of Utah, 1930, 15 pp.

Leaver, E. S., and Woolf, J. A. In Reply to Communication of E. H. Hamilton on Paper on Metallurgy of Mother Lode of California Carbonaceous Ores. Eng. and Min. Jour., vol. 130, 1930, pp. 32-33.

________________ Retreatment of Mother Lode Carbonaceous Slime Tails. Rept. of Investigations 2998, Bureau of Mines, 1930, 6 pp.

________________ Retreatment of Mother Lode (California) Carbonaceous Slime Tailings. Tech. Paper 481, Bureau of Mines, 1930, 20 pp. 5 cents.

<u>Nonferrous Metals</u>:

Work on lead, zinc, copper, and quicksilver has been continued. The work on the fundamentals of zinc metallurgy has been brought to completion and summarized in a bulletin by C. G. Maier.

Maier, C. G. Zinc Smelting from a Chemical and Thermodynamic Viewpoint. Bull. 324, Bureau of Mines, 1930, 93 pp. 20 cents.

________ The Present Status of our Quicksilver Industry. Trans. Am. Inst. of Min. and Met. Engrs., 1930, pp. 299-312. Am. Inst. Min. and Met. Eng., Tech. Pub. 264, 1930, pp. 19-32.

<u>Laboratory Apparatus and Methods</u>:

In the course of its laboratory work, new devices are frequently evolved by the bureau. Three such improvements have been described by Maier this year and one by Head and Slavin.

Head, R. E., and Slavin, M. A New Development in the Preparation of Briquetted Mineral Grains. Coop. Tech. Paper 10, Bureau of Mines and University of Utah, 1930, 11 pp.

Maier, C. G. Magnetic Switches in Regulatory Circuits. Ind. and Eng. Chem., Analytical Ed., vol. 2, 1930, p. 258.

________ Resistance Thermometers for Chemists. Jour. Phys. Chem., vol. 34, 1930, p. 2860-2868.

________ Cement for Silica-Glass Joints. Ind. and Eng. Chem., Analytical Ed., vol. 2, 1930, p. 337.

<u>Discussions</u>:

The following discussions of papers in the technical press have been published:

Maier, C. G. Correspondence on Page 402 of April Issue of Industrial and Engineering Chemistry Concerning Question of Decomposition of Steam by Metallic Copper. Ind. and Eng. Chem., vol. 22, 1930, p. 916.

__________ Discussion of Electrolytic Iron from Sulphide Ores, by R. D. Pike et al. Trans. Iron and Steel Div. of Am. Inst. Min. and Met. Eng., 1930, p. 344.

__________ Discussion of Am. Inst. Min. and Met. Engrs. Tech. Pub. 270, A Theory Concerning Gases in Refined Copper, by A. E. Wells and R. C. Dalzell. Trans. Am. Inst. Min. and Met. Eng., Institute of Metals Division, 1930, pp. 364-366.

A limited number of most of the Bureau of Mines reports of investigations and information circulars are available for free distribution. If you wish to obtain copies of such publications, address the Publications Section, U. S. Bureau of Mines, Washington, D. C. Bureau of Mines bulletins and technical papers can be obtained only from the Superintendent of Documents, Government Printing Office, Washington, D. C., at the prices indicated. Cooperative papers of the Bureau of Mines and a university must be obtained through application to the cooperating university. As no provision is made for supplying the Bureau of Mines with reprints of articles appearing in the technical press, such articles will have to be consulted in the journals in which they appear.

I. C. 6450

June, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MINING LAWS OF HUNGARY

BY

E. P. YOUNGMAN

I. C. 6450.
June, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING LAWS OF HUNGARY[1]

By E. P. Youngman[2]

PREFATORY NOTE

This paper is one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the laws of Hungary was prepared from information furnished by Nicholas Roosevelt, American minister at Budapest, in response to a questionnaire submitted by the United States Bureau of Mines and transmitted through the courtesy of the Department of State. Additional information was obtained from W. A. Hodgman, American commercial attaché, also at Budapest, whose report was made available through the courtesy of the commercial laws division of the Bureau of Foreign and Domestic Commerce.

INTRODUCTION

The mining laws referred to by those who furnished the reports mentioned in the preceding paragraph are: General Austrian Mining Law of 1854 and the Law of 1868, which prescribes for Hungary (as well as for Transylvania) the full text and decisions of the General Austrian Mining Law; Government Decree 18346, 1859; Parliamentary Decree of 1861; the Law of 1871; the Commercial Code, Law 37 of 1875; the Industrial Law of 1884; the Law of 1886; the Decree of the High Court of Justice, 5540, of 1899; Laws 6 and 7 of 1911; Law of 1921; Laws 23 and 24 of 1922; Laws 5 and 21 of 1927; and Law 40 of 1928. (Law 21 of 1927 and Law 40 of 1928 are labor laws.)

A compilation of the mining laws of Hungary, "Magyar Banya Jog," or the "Hungarian Mining Law," was made use of in part. This volume is the second edition of a school textbook, which was prepared by Dr. Bela Balkay and Dr. Imre Szeoke, and which was published by the Apollo Press (V., Sas-utca 13, Budapest) in 1909.

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1 The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6450."

2 Rare metals and nonmetals division.

RIGHTS OF FOREIGNERS

Americans, as well as all other foreigners, are permitted to explore, and to own, and operate mines in Hungary on the same terms as those prescribed for Hungarian citizens, except that a foreigner must appoint an Hungarian national as his agent and must report his name to the mining authority having "first-instance jurisdiction" in the district in which the mine is situated, in which district the agent must reside.

The law granting equal rights to all in connection with exploration and with the ownership of mines is comprised in paragraphs 7 and 8 of the Parliamentary Decree of 1861. Through the Commercial Law of 1875 (par. 210), also, foreigners are given equal rights with Hungarians through the permission granted to them to take part in the commerce and industry of Hungary (as mining in general relates to industrial production).

Foreigners are liable, with respect to mining enterprises, to the same taxes and duties as are Hungarian citizens.

An individual or private owner need not incorporate; but firms must incorporate, according to article 210 of the Commercial Code (Law 37 of 1875). A foreign corporation must have an agency on the property, and this agency must be registered with the local court of justice.

The law does not designate that a certain percentage of ownership shall rest with Hungarian nationals.

CLASSIFICATION OF MINERALS

According to article 3 of the Hungarian Mining Law, minerals are classed as "free," "monopolistic," and "those that may be mined with the permission of the owner."

"Free minerals" are those that "may be used because they contain metals, sulphur, alum, iron or copper sulphate, liquid sulphates, graphite, and bauxite." These may be searched for freely--that is, under permits obtained from local mining authorities.

"Monopolistic minerals" include common salt (which is under exclusive Government monopoly), potash (kali salt), natural gas, mineral oil, and "ozokerite" (ozocerite). These may be explored under a special agreement with the State, previously obtained.

"Minerals that may be mined with the permission of the owner" include all not classed as "monopolistic" or "free." Their extraction falls not under mining but under trades and is subject to the provisions of the Trade Law.

Coal, which may not be mined except with the permission of the surface owner, is governed by special legislation. (See section of this paper entitled "Special Provisions with Regard to Coal.")

OWNERSHIP OF MINERALS

As has been indicated under the section of this paper entitled "Classification of Minerals," the three general principles of ownership of minerals are involved in the mining laws of Hungary, as follows:

<u>Regality</u>.- The State owns salt (common and potash), natural gas, mineral oil, and ozocerite (monopolistic minerals).

<u>Res nullius</u>.- "Minerals that may be used because they contain metals, sulphur, alum, iron or copper sulphates, liquid sulphates, graphite, and bauxite" (free minerals) may be mined by any one, provided he has obtained a permit from the mining authorities and complied with the other provisions of the law.

<u>Common-law theory</u>.- All minerals not classed as "monopolistic" or "free" are the property of the owner of the soil, who may bargain for their exploration and exploitation, under the provisions of the Trade Law.

PROSPECTING AND MINING PERMITS

A prospecting permit may be general (preliminary) or exclusive. A general permit is issued for the entire district over which the mining authority "of first instance" has jurisdiction. An exclusive permit covers a circle having a specified radius and having its center at the point where the mineral has been discovered. (Art. 14, Hungarian Mining Law.) A prospecting license is valid for one year from the day upon which the claim is filed, and it may be renewed from year to year. (Art. 16, Hungarian Mining Law.)

A permit to mine, called "a mining-lease donation," is granted by a donation act of the Government. (Art. 40, Hungarian Mining Law.) It conveys the exploring right and even the ownership of the mine. This ownership corresponds fully with the ownership provisions of the Private Law, except that mine grants may be cancelled by the authorities. The duration of a grant for the mining of "free minerals" and coal is indefinite--as long as the minerals are mined. Such a grant, of course, does not require renewal. A concession to mine the minerals reserved to the State may be limited by the agreement between the State and the grantee; likewise, renewal depends upon the Government.

MINING AUTHORITIES

Divisional mine inspectors (mine captaincies) have the authority, in the first instance, to issue prospecting permits; they likewise grant mining permits, except in the case of the so-called "monopolistic minerals," which are under the authority of the Ministry of Finance, which is the highest mining authority.

Mining authorities do not have the right to refuse a permit to an applicant that has complied with the requirements of the law.

The duties and the powers of mine officials are contained in article 31 of the Law of 1871.

Eight courts of justice, endowed with the duties of mine tribunals, in the several divisions of the country have supervision over mining authorities and mining matters. (Art. 31, Law of 1871.)

SIZE OF PROSPECTING AND MINING AREAS

A general exploring permit covers a whole mining district (see section of this paper entitled "Permits"). An exclusive exploring permit covers a circle having a radius of 424.812 meters and having its center at the point where the mineral has been discovered and reported by the prospector. The number of circles that may be granted to one prospector is not limited. (Art. 19, Hungarian Mining Law.)

A mining area for a single working of coal, natural gas, mineral oil, and ozocerite may be 360,931.2 square meters; for the "free minerals" the area may be 180,465.6 square meters. (Art. 42 and 47, Hungarian Mining Law.)

QUALIFICATIONS OF PERMITTEES

The legal requirements of applicants for ownership, possession, or lease are almost the same as those prescribed in the Hungarian Private Law. An age limit is not prescribed, although minors may explore or own mines only through their legal guardians. The amount of capital is not specified. Paragraphs 218 and 219 of the General Austrian Mining Law decree that the operator must have a commercial representative endowed with full responsibility in the place where the mining is being carried on. An expert engineer or mine specialist must be provided to supervise the work.

PRIORITY RIGHTS

The explorer of "free minerals" by prospecting acquires the exclusive priority right to obtain a mining concession. A license to mine coal may be granted only with the consent of the owner of the land. The license

to mine potash salt, natural gas, mineral oil, and ozocerite may be granted only in the name of the State, but the Government may give to others the right to mine these so-called "monopolistic minerals."

RIGHTS AND OBLIGATIONS OF PERMITTEES

A license gives a prospector a personal and exclusive right, which the landowner may not refuse. If the prospector can not come to an agreement with the surface owner concerning the occupation of the land necessary for prospecting, article 98 of the Mining Law entitles him to apply for expropriation rights. In the case of expropriation, the authorities fix the amount of yearly rental and damages. (Otherwise the contract between the landowner and the permit holder settles the sums to be paid for rental and damages.) Land can not be fully expropriated against a lump-sum payment except with the owner's consent. (Art. 100, Hungarian Mining Law.)

Without the owner's permission, no prospecting is allowed within certain distances of dwellings or other buildings or within courtyards or other fenced-in territories. (Art. 17, Hungarian Mining Law.)

A permit is required to prospect within certain distances of cemeteries, public roads, railroads, waterworks, and frontiers. (Art. 18, Hungarian Mining Law.) A special permit or agreement with respect to the use of roads, water, et cetera, is required, the points of agreement being covered in the official license.

To a landowner requesting it, a prospector must furnish security for the first 30 days of prospecting. (Art. 43, Law of 1868.)

Personal differences and protests are under the authority of the Private Law.

WORK REQUIRED

The law requires exploration work to be continuous; failure to carry on the work for one year may result in the cancellation of the permit. (Art. 180 to 182 of the General Austrian Mining Law.) If the prospector has the right to search for minerals in several circles, and if similar deposits occur in all the search circles, he may concentrate the work into one circle, with the permission of the mining authorities. (Art. 174, Hungarian Mining Law.)

TRANSFERS

The right to mine coal and the "free minerals" may be transferred, with the approval of the mining authorities, provided the interested persons are legally competent; but the concession granted for the mining of "monopolistic minerals" (see section of this paper entitled "Classification of Minerals") may be transferred only with the permission of the State--that is, of the Hungarian Parliament. (Art. 122, Hungarian Mining Law.) Changes in leasehold or ownership must be entered in the public register of mining areas. (Art. 108 to 133 of the General Austrian Mining Law.)

CANCELLATION

A grant for the mining of free minerals and coal may not be cancelled by the mining authorities except when the owner fails to comply with the provisions of article 174 of the Mining Law, which stipulates that mines must be worked continuously unless suspension of work is sanctioned by the mining authorities.

Cancellation of a State concession depends upon the terms of the agreement between the Government and the grantee.

SPECIAL PROVISIONS WITH REGARD TO COAL

The first Government decree concerning coal, which was issued in 1861, declares that coal may not be explored on private land without the consent of the surface owner, even when there has been a change in the ownership of the land. Paragraphs 284 and 285 of the General Austrian Mining Law confirm this decision and obligate the explorer to pay a low rental to the surface owner.

Decree 5540, 1899, of the High Court of Justice provides that a coal-mine lease shall include the rights of real estate because of the close contact between land and subsoil. But if the license contains no agreement between owner and explorer concerning possession, leasehold, and related matters, the coal license is protected by law in the same way that any movable object is.

For the Government's levy on coal, see section of this paper entitled "Fees, Rents or Royalties, and Taxes."

FEES, RENTS OR ROYALTIES, AND TAXES

The general Government fee for each exclusive "search-circle" prospecting area is 11 pengő[3] annually; for each mining claim it is 22 pengő.

The yearly inspection fee for a single working area of coal, natural gas, mineral, and ozocerite is 176 pengő; for sulphur, alum, iron or copper sulphate, liquid sulphates, graphite, and bauxite, the fee for a single working area is 88 pengő. (Art. 42 and 47, Hungarian Mining Law.)

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3 The value of the pengő in United States currency in 1929 was 17.4414 cents. The pengő is a new unit (subdivided into 100 filler), adopted by the Hungarian Government on Nov. 4, 1925, and declared in force on Dec. 27, 1926. (The pengő was equal to 12,500 paper crowns.)

No rent, or royalty, is due for the mining of the "free minerals." With respect to coal, a royalty is due to the surface owner; the amount depends upon the agreement between the interested persons. The usual royalty is from $1\frac{1}{2}$ to 5 per cent of the sales price at the pit, depending upon the quality of the coal. The amount of rent due for a State concession also is governed by a personal agreement between the landowner and the explorer, in accordance with the Private Law.

Additional rentals are levied on coal lands (between 16 and 30 filler a unit), according to article 17, Law of 1921, and article 5, Law of 1927.

A private owner, if not incorporated, pays a 5-per cent earnings tax, in addition to a personal income tax, graduated from 1 to 40 per cent of his net income. An incorporated firm pays a 5-per cent earnings tax and a 5 to 25 per cent corporation tax. (Laws 23 and 24 of 1922 and Law 5 of 1927.)

PENALTIES

The criminal section of the law provides for the punishment of any one prospecting without a permit, exploring in an area not granted, exploring with a permit when a donation act is required, exploring before a license has been received, or transferring work to another area.

MONOPOLIES

Common salt seems to be the only mineral that is the subject of an absolute Government monopoly. All private undertaking is forbidden in salt mines. The section of the law that places minerals in different categories lists mineral oils, natural gases, potash salts, et cetera, as monopolies. Nevertheless, provision is made elsewhere for the granting of concessions to private persons for the exploring of these minerals.

MISCELLANEOUS

The Industrial Law of 1884 (art. 17) governs the industrial side of mining, such as the establishing of plants, machine houses, timber works, et cetera.

The labor laws require risk, accident, and old-age insurance. (Law 21 of 1927 and Law 40 of 1923.)

There are no export duties, and there is no sales control. Special railway rates may be granted.

With respect to juridical matters, mining enterprises and trade companies come under the same provisions.

I. C. 6451

June, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MINING LAWS OF THE NETHERLAND EAST INDIES

BY

E. P. YOUNGMAN

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE--BUREAU OF MINES

MINING LAWS OF THE NETHERLAND EAST INDIES[1]

By E. P. Youngman[2]

PREFATORY NOTE

This paper presents one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the rights of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the laws of the Netherland East Indies has been prepared from translations of the mining laws and ordinances now in force in that country, submitted by Coert du Bois, American consul general, at Batavia, Java, and obtained through the commercial laws division of the Bureau of Foreign and Domestic Commerce. This abstract has been checked against the answers made by Hallett Johnson, American chargé d'affaires, at The Hague, to a questionnaire prepared by the Bureau of Mines and transmitted through the courtesy of the Department of State.

INTRODUCTION

The basic mining law of Netherland India is contained in N. I. Statute 1899, No. 214, as amended by N. I. Statute 1910, No. 588, and 1919, No. 4, which is quoted as the "Indian Mining Law," the effective date of which was fixed as May 8, 1907, by proclamation dated October 12, 1906. The Indian Mining Law was supplemented by Ordinance No. 38, of 1930, which was passed by the Volksraad on March 18, 1930, to become effective on October 1, 1930 (superseding an earlier ordinance entitled "Government Gazette," 1906, No. 434). An important law with respect to American petroleum rights, Statute No. 23, passed February 9, 1928, approving contracts between the Indian Government and the Netherland subsidiary of a large American oil company, is in the nature of a private law or of an enabling act (see section of this paper entitled "Rights of Foreigners"). Statute 1930, No. 348, effec-

1 The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6451."

2 Rare metals and nonmetals division.

tive October 1, 1930, sets forth in minute detail conditions under which prospecting and mining permits may be granted. Statute 1930, No. 341, effective October 1, 1930, embodies the police regulations of the mining industry.

All citations in this abstract designated by numbers alone refer to the Indian Mining Law of May 8, 1907. "M. O. 38" refers to the supplementary ordinance of 1930. "S. 23" refers to the petroleum law of February 9, 1928. Mere mention is made of the other statutes enumerated in the foregoing paragraph.

MINERALS SUBJECT TO THE INDIAN MINING LAWS

The following minerals, designated in the Indian Mining Law (sec. 1, art. 1) as those that the holders of surface rights may not dispose of, are governed by this law and supplementary ordinances:

(a) Precious stones, graphite, platinum, osmium, iridium, gold, silver, mercury, bismuth, molybdenum, tin, tungsten, lead, copper, zinc, cadmium, nickel, cobalt, chromium, iron, manganese, antimony, arsenic, and strontium, either native or as ore, and also other minerals if they are found in the same load and "thus have to be exploited together"; minerals that may be exploited on account of their sulphur content (including the making of alum); phosphates that may be used as fertilizers; saltpeter; rock salt[3] and all salts that are associated therewith in the same deposit.

(b) Anthracite and all kinds of pit coal and brown coal (lignite); petroleum, asphaltum (mineral pitch), resin (mineral wax), and all other kinds of bituminous substances, both solid and liquid, and inflammable gases (except marsh gases); iodine and its compounds.

The final decision as to whether a mineral belongs to the classes mentioned in the preceding paragraphs rests with the governor general. (Sec. 2, art. 1.)

RIGHTS RESERVED TO THE GOVERNMENT

Article 5a of the Indian Mining Law entitles the Government to have prospecting and developing carried out wherever such work would not interfere with rights granted to permit holders or concessionaires. The Government may do the work itself or may enter into agreements with persons or companies that have complied with the provisions of the law with respect to citizenship or domicile. Such agreements, unless they refer exclusively to prospecting shall not be entered into without authority of law,

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3 Rock salt, however, can not be mined except at places not subject to the salt monopoly. (Art. 5.)

however. The decree reserving areas or parts of the country must mention the minerals to be prospected for or mined. Article 23, in section 2a, provides that petroleum and allied minerals, which come under subsection (a), section 1, article 1 (see p. 1 of this paper), may be mined only by the Government or by persons or companies under contract with the Government.

RIGHTS OF FOREIGNERS

The language of the law itself does not expressly discriminate against foreigners, although, as in many other countries, foreigners must obtain domicile rights, and companies not having their seat in the Netherlands or the Netherland Indies must have their representatives there. The full language of the law in this respect, as set forth in article 4 of the Indian Mining Law, is given in a succeeding paragraph.

With respect to petroleum rights, at least, actual reciprocity between the Netherland Indies and the United States has recently been clarified. Although the terms of the law seemed to permit the acquiring of mineral leases or concessions affecting mineral lands in the Netherlands or her colonial possessions by American-owned corporations formed under the laws of the Netherlands or of her possessions, repeated applications by or on behalf of American interests had been rejected, although the Netherland Government had enjoyed a similar right in the United States for a number of years (since 1923). As a result of diplomatic exchanges between the two Governments, the law of February 9, 1928 (S. 23) was passed, authorizing the Minister of Colonies, as representing the Netherland Indies, to sign contracts with a company in which American capital is heavily interested, making it possible for that company to exploit important petroleum lands in the Netherland Indies. In return, the United States Government formally recognized the Netherlands as a reciprocating country within the meaning of the United States Mineral Leasing Act of February 25, 1920.

Although the law itself does not discriminate against foreigners, and article 5a of the Indian Mining Law expressly authorizes the Indian Government to reserve lands and to enter into contracts with persons or companies for the exploration of the land and the exploitation of the minerals reserved to itself (as are petroleum and allied substances), the law further provides, in the same article, that legislative approval must be obtained of any contracts so formed; therefore, American rights with respect to minerals reserved to the Government depend in each instance upon satisfactory negotiations with the Government of the Netherland Indies and also upon favorable parliamentary action.

Article 4 of the Indian Mining Law reads as follows:

1. No one may hold prospecting licenses or concessions, except:

    a. Full citizens of the Netherlands.

b. Those domiciled in the Netherlands or the Netherland Indies and who have the rights of domiciled persons.

c. Companies, which have their seat in the Netherlands or the Netherland Indies, and, if limited liability companies, whose manager or director (or if there are two, both, or if there are more managers, the majority, as well as the majority of the directors) and if firms (partnerships), whether with working or sleeping partners, whose managing partner (or if there are two, both, or if there are more managing partners, the majority of them) are full citizens of the Netherlands or are domiciled in the Netherland Indies and have full rights of domiciled persons and, if the latter, at the same time reside in the Netherland Indies or in the Netherlands,--

On condition that persons that are not residing or companies that have no seat in the Netherland Indies shall be obliged to have a duly appointed representative there; and that persons residing in the Netherland Indies, being the representatives in the Netherland Indies of those who are not residing in the Netherland Indies, and the directors who are residing in the Netherland Indies or the representatives of companies that have their seat in those Indies or in the Netherlands be entitled to reside within the province or provinces in which the prospecting license or concession has been granted.

2. Applicants for a prospecting license or a concession shall be obliged to fix their (legal) domicile during the life of the license or the concession and for all matters connected therewith at the head office of the provincial Government within which the prospecting area or the concession area is wholly or partly situated.

3. The rights and obligations connected with a prospecting license, as well as those connected with a concession, revert from the legal possessor at his death to his rightful successor, if they either immediately or within a year from the date when the inheritance became available comply with the provisions of this article. They may be assigned within the year to persons or companies that comply with the provisions of this article and in case of prospecting licenses observe the stipulations of section 7 of article 7 and for the rest on condition of sanction's being obtained from the governor general.

4. Differences as to whether the provisions of this article have been complied with shall be decided by the judge in the manner laid down by ordinance.

The following paragraph from a memorandum, dated July 10, 1928, addressed to the American Minister at The Hague by the Dutch Minister for Foreign Affairs, sets forth the official attitude of the Netherlands:

Her Majesty's Government has no intention of abandoning the open-door policy in so far as the granting of rights for the exploitation of oil lands in the Netherland Indies is concerned; consequently, the opportunity will remain open for American interests to participate in the exploitation of the petroleum wealth of the Netherland Indies. It is, however, understood that this policy does not imply that in each specific instance of the granting of petroleum rights to other than American interests the question of the granting of identical rights to American interests can be raised.

One condition of a permit or a concession may be that the personnel working on the estate must consist partly or entirely of Netherland subjects. (Art. 70, M. O. 38.) According to Hallett Johnson, American chargé d'affaires at The Hague (January 23, 1931), the usual requirement is that two-thirds of the employees must be Netherland nationals.

OWNERSHIP

The general principles of the ownership of minerals, as described in mining legislation, are as follows:

1. Regality, according to which minerals belong to the State, which has the exclusive right to extract them or have them extracted.

2. Common-law theory of ownership, whereby the owner of the surface land owns also the subsoil.

3. Res nullius, the theory that the minerals are owned by no one and that the rights of disposal belong to the discoverer, within limits imposed by law.

In the East Indian Mining Law applications of all three theories are found, varying with the nature of the mineral, as well as with the reservation or nonreservation of the land.

The right of the Government to reserve land and control all minerals thereon (which is exercised fully in the case of petroleum) is an example of the theory of regality. The common-law theory of ownership (adopted in the United States) has but slight application in Netherland Indies. Subsection (a) of article 6 of the Indian Mining Law exempts from the provisions of the law minerals exploited by natives (or those placed on the same footing) if the mining is done on a small scale and for their own account and profit. The third theory (res nullius), however, is the one generally applicable to minerals other than coal and petroleum, as any one, by complying with the conditions of the law, may obtain prospecting and mining permits, and the owner of the surface may not prevent such operations on his land by a licensee of the Government.

PROSPECTING

Application.--An application for a prospecting permit is made to the head of the Mining Service, who has power to grant or refuse the petition, but whose decision is subject to appeal to the governor general of the East Indies. The application must be in the form prescribed by the head of the Mining Service. It must be accompanied by an unstamped copy, certified by the applicant, as well as by a map in duplicate and by any other required documents. (Art. 12, M. O. 38). The chief of the Mining Service shall place on both copies of the application the date and the hour of their receipt and shall return one copy to the applicant. (Sec. 2, art. 7.) Every applicant for a permit must present documentary evidence that he has complied with the requirements of article 4 of the Indian Mining Law, with respect to citizenship or domicile rights. (Art. 13, M. O. 38.) The chief of the Mining Service publishes in the Javasche Courant the hours during which daily (except Sundays and holidays) applications will be received by him or by another officer appointed for that purpose (whose appointment also is published in the Javasche Courant). Applications received at hours other than those designated or those received by mail are considered as having been submitted at the beginning of the following series of office hours. Applications received in the course of the same hour are considered as having been submitted simultaneously. (Art. 10, M. O. 38.)

The right to prospect a given area may be granted to only one person or one company. (Art. 77, M. O. 38.)

Priority.--The application that has been handed in first, in order of date, shall have preference. Should two permits be asked for simultaneously, for the same area, the drawing of lots decides which application shall have preference. (Sec. 3, art. 7; sec. 1, art. 22, M. O. 38.) (For further details, see sections 2 to 6, inclusive, art. 22, M. O. 38.) Appeals from decisions upon applications may be made to the governor general, who may set aside, on the ground of equity, the provisions with respect to the time of the receipt of applications. (Sec. 5, art. 7.)

Protests or objections.--Those who have a right to the land, interested third parties, and all whose interests might be endangered by the granting of a permit may submit their objections to the head of the provincial administration, as well as to the head of the local administration, within one month after the application for a permit has been announced and within two days from the day on which the application has been published in the Javasche Courant. Natives and foreign orientals may lodge their protests also with the nearest proper European official and with any native official that may have been detailed for that purpose by the head of the local administration. Natives and foreign orientals may present their objections verbally, whereas all others must make their protests in writing. (Sec. 1, art. 21, M. O. 38.) Further details are given in sections 2 to 6 of article 21 of Mining Ordinance 38. The applicant for a permit is informed by the head of the Mining Service of any objections that have been raised. (Sec. 5, art. 21, M. O. 38.)

Decree granting permit.--A certified copy of the decree granting a permit is issued to the applicant. No charge is made other than stamp duty for the first copy. (Sec. 1, art. 26, M. O. 38.) In the decree granting a permit, mention must be made of any objections that may have been made, or specific mention must be made of the fact that no objections have been raised. A decree refusing a permit must contain the reasons for refusal. (Sec. 1 and 2, art. 24, M. O. 38.)

Conditions may be attached to the license. (Sec. 4, art. 7.) Statute 1930, No. 348 (October 1, 1930) contains the most recent conditions imposed upon the granting of a prospecting permit. They relate to the furnishing of information with respect to compliance with the mining laws; the supplying of data requested in an investigation; the presenting by heirs of proof of compliance with article 4 of the Indian Mining Law; the reporting of changes in personnel of companies; the recording and reporting of mining activities and mineral production; the making by the Government of appraisals of taxable commercial products; and like matters. These conditions may not be altered or increased in number so long as the permit is in force, except with the approval of the holder thereof. (Sec. 1, art. 69, M. O. 38.) No new or altered conditions may be attached to the extension of a permit, to the license to transfer a permit, or to a new permit, except with the consent of the holder thereof or of the party that obtains it. (Art. 71, M. O. 38.)

Publication.--The head of the Mining Service, after investigation, announces in the next issue of the Javasche Courant the application that may be granted, sending a copy of the application (with a map or drawing) to the head of the provincial administration at whose office the applicant has taken his domicile rights, who in turn publishes the application locally as soon as possible. If the area applied for is located in more than one province, the head of the provincial administration, at whose office the domicile rights have been taken, consults his colleagues before publishing the application. If settlements are located in the area applied for, the application is announced verbally in the native language by the head of the local administration or by another officer appointed for that purpose. A notice is posted in Malay, in the local native language, and, if necessary, in Dutch. (Sec. 1-3, art. 20, M. O. 38.) A decree granting an application for a permit or one approving a transfer must be published in the Javasche Courant. (Sec. 3, art. 24, M. O. 38.)

Area and boundaries.--A prospecting license shall be for a field of indefinite depth, vertically within the limits of the area set forth in the deed of license. (Sec. 4, art. 7.) An application for a prospecting permit may concern only one area forming a coherent unit. (Art. 17, M. O. 38.) An area for which a permit is issued may not be larger than 10,000 hectares, unless the head of the Mining Service, in agreement with the head of the provincial administration concerned, consents to a larger area. Should these authorities disagree, the head of the Mining Service applies to the Director of the Department of Government Industries for a decision. (Sec. 1-2, art. 19, M. O. 38.)

The corners of an area applied for must be chosen in such a way that at least one of them is either a permanent point that can be easily located in the field and not subject to change or removal or one that may be determined easily in relation to such a point in the vicinity. The connecting lines between the corners must be straight lines, as far as possible. Lines of which the azimuth is not known may not be used as border lines. Meridians and parallels, as well as lines forming angles with them, may be used as border lines, provided that they are determined by points in the vicinity of the area. Clearly distinguishable natural borders may be used as border lines. River banks may be used only when the course of the river is sharply defined and does not run through marshy districts. Likewise, borders between provinces, regencies, divisions, districts, and like territories may be used. (Art. 18, M. O. 38.)

Areas and boundaries may be changed by a new decree at the discretion of the head of the Mining Service upon further information obtained after the issuance of the original decree. The decision of the head of the Mining Service is final with respect to boundaries. (Art. 55, M. O. 38.)

Requirement respecting work.--The only apparent requirement with respect to work to be done under a prospecting license is that prospecting must be commenced within a period of one year from the date of the granting of the license. (Sec. 6, art. 7.) The decision as to whether exploration has been commenced within the required period rests with the head of the Mining Service. (Art. 57.) Waiver of the time limits may be made by the governor general should it be impossible to commence exploration or be necessary to stop exploration partly or entirely because of controlling forces not included in the ordinary risks. (Art. 65, M. O. 38.)

Duration of permit.--A prospecting license shall be granted for a certain period, not to exceed three consecutive years. It may be renewed twice, by the head of the Mining Service, each time for a maximum period of one year, and upon an application of the licensee made within three months before the end of the period for which the license was granted. (Sec. 4, art. 7.) Applications submitted after or more than three months before the expiration of the original permit are rejected unreservedly. (Art. 61, M. O. 38.)

Transfer.--A prospecting permit is transferable only with the consent of the Government. (Sec. 7, art. 7.) Permission to transfer is refused when it has not been proved (by submitting receipts or in any other way) that amounts due for taxes and permanent duty have been paid, or when the prospecting permit has been cancelled. (Art. 25, M. O. 38.)

A certified copy of the decree granting a transfer is issued to each of the parties concerned in the transfer, for which no other charge than stamp duty is made. (Sec. 1, art. 26, M. O. 38.)

A permit may be transferred to only one person or one company. (Sec. 1, art. 77, M. O. 38.) A permit may not be transferred to (a) minors, (b) persons of unsound mind, or (c) limited companies the activities of which, according to the statutes, do not include mining operations. (Sec. 2, art. 77, M. O. 38.)

Termination of permits.--A prospecting license is terminated by law (a) at the end of the period for which it is granted or renewed, (b) if the holder thereof fails to comply with the provisions with respect to citizenship or domicile rights, and (c) if, in the event of the death of the rightful holder, his successors do not comply with the provisions mentioned under (b) within the period allowed. The license shall be cancelled (a) if prospecting operations are not commenced within a period of one year from the date of the granting of the license, and (b) at the request of the holders of surface rights or of interested third parties if operations have been undertaken without the holder's having complied with the stipulations contained in section 9 of the Mining Law with respect to notifying the surface owner of the intention to prospect and with respect to damages. (See section entitled "Obligations and Rights of Surface Owners" of this paper.) The license may be cancelled (a) if the licensee fails to comply with any of the conditions under which the license was granted, or (b) at the petition of the licensee, either for the whole or for a part of the prospecting area. In the latter case, the license holder shall present his petition, in duplicate, to the proper official, who shall write upon both copies the date and the hour of the receipt of the petition, and who shall return one copy to the petitioner, and who shall render a decision within three months.

The cancelling of a permit shall be done by the authority by whom the license was granted, but not until the holder has had an opportunity to defend his interests. Appeal from the decision with respect to cancellation may be made to the governor general, in the manner prescribed by ordinance. Decrees cancelling permits must mention the grounds upon which cancellation is based and must be served legally through the offices of the head of the Mining Service, unless the cancellation has been made at the request of the holder.

Cancellation of a permit at the request of the holder goes into effect as soon as the decree has been issued to him. In all other cases, cancellation goes into effect on the day that it becomes irrevocable. (Art. 11-12; art. 58-60, M. O. 38.)

Rights of the prospector.--The permit grants to the holder thereof, to the exclusion of all others, the right to perform all the work necessary on the prospecting field for the finding of the minerals named in the section entitled "Minerals Subject to the Indian Mining Law," or for the purpose of ascertaining the nature of the mineral reefs discovered and of the minerals found therein, according to the provisions of the act and the conditions under which the license was granted, on condition that the area has not been reserved for those minerals or a concession granted therefor or an agreement entered into concerning them. (Sec. 1, art. 10.)

CONCESSIONS

Mining concessions are granted only for the mining of minerals mentioned in article 1 of the Indian Mining Law (see section of this paper entitled "Minerals Subject to the Indian Mining Law") whose existence in the mining field as a natural deposit for which their extraction is technically possible has been proved to the satisfaction of the governor general. (Sec. 2, art. 13.) They shall be granted only to applicants who have complied with the provisions concerning citizenship and domicile (art. 4.), as given in the section of this paper entitled "Rights of Foreigners."

Application.--The requirements with respect to applications for mining concessions are the same as those for applications for prospecting permits (see foregoing section entitled "Prospecting"), as far as the authority to whom they are presented and the rulings with respect to the time of their receipt are concerned. (Art. 27, M. O. 38.) However, the governor general is the official empowered to grant or refuse a mining concession. (See section of this paper entitled "Decree of Concession.")

A petition for a concession must state:

1. The name (Christian name and surname) of the petitioner and his place of residence. (Sec. 5, art. 28.)

2. The profession and the age of the applicant. (Sec. 1, art. 29, M. O. 38.)

3. The name of the mineral, or minerals, for which the concession is requested. (Sec. 5, art. 28.)

4. Whether the discovery has been made by the applicant as the holder of a permit or as a concession holder, with a notation of the date and the number of the decree granting the permit or the concession. (Sec. 2, art. 29, M. O. 38.)

5. The location of the discovered deposit or deposits and the boundaries of the area either claimed for concession or already granted as such. (Sec. 5, art. 28.)

6. The location of the native clearances that may be within the limits of the area applied for and the border of the area or areas within which the applicant may not carry out any clearing work without the permission of those who have the right to the native clearings lying therein. (Sec. 3, art. 29, M. O. 38.)

7. The name to be given or already given to the concession. (Sec. 5, art. 28.)

8. The chosen domicile of the petitioner. (Sec. 5, art. 28.)

Supplementary documents required (art. 30, M. O. 38) are:

1. Evidence that the minerals sought exist in a workable deposit.

2. A map, in triplicate, of the area applied for, on a scale not smaller than 1 to 25,000, by a sworn surveyor (or by a person acceptable to the head of the Mining Service), on which the following facts must be indicated:

(a) The borders of the area applied for.

(b) The points suitable for placing signs, two consecutive ones not to be farther apart than 500 meters.

(c) The location of the deposit or deposits discovered.

(d) The location of the native clearings that may be within the borders of the area applied for and the borders of the area or areas within which the concession holder is not allowed to carry out any clearing work except with the consent of those having a right to the native clearings situated therein.

(e) The natural or artificial distinct permanent points on the surface for the purpose of finding bearings.

(f) The astronomical meridian.

(g) The borders of the area covered by a permit or of any part thereof. (This need be included only upon demand by the head of the Mining Service.)

Modifications of or supplements to the foregoing requirements are set forth in sections 2 to 5, inclusive, of article 30, of Mining Ordinance 38.

Conditions of law with respect to overlapping fields are covered by article 46 of Mining Ordinance 38.

The right to a concession or a concession may be granted to only one person or one company. (Sec. 1, art. 77, M. O. 38.)

Right of priority to the discoverer.--The discovery of a mineral mentioned in the first paragraph of the section of this paper entitled "Minerals Subject to the Indian Mining Law" gives the right to the discoverer who is either the owner of a prospecting license or a concessionaire on that particular field and who has demonstrated that certain minerals exist and that their acquisition is technically possible to obtain a

concession to mine the discovered mineral, provided he maintains his claim within the period for which the license was granted or an extended period and hands in his petition first in order of date, and provided further that the governor general does not decide that the exploitation of the mineral would not be for the general interest. Should applications for the same field be submitted simultaneously, the decision depends upon the drawing of lots, according to the regulations of section 1, of article 22, of Ordinance 38. (Sec. 1-2, art. 28; art. 28, M. O. 38.)

The mining of minerals mentioned in paragraph 2 of the section entitled "Minerals Subject to the Indian Mining Law" (coal, petroleum, and all kinds of bituminous substances) may be done only by the Government or by virtue of an agreement with the Government. Although the demonstration of the presence of these minerals does not give the right to a concession, it may give the right to a reward. The reward for the discovery of these minerals is determined by the governor general. (Sec. 2a, art. 28.)

Objections.--Concessions are granted only after all interested parties have had an opportunity to defend their interests (art. 14), which they may plead within one month after an application for a concession has been announced and within two months after the announcement has appeared in the Javasche Courant. They may raise their objections before the head of the local government and, where natives and foreign orientals are concerned, also before the nearest proper European officer and such native officer as may be detailed for that purpose. Natives and foreign orientals may submit their objections orally, as well as in writing, the oral objections being officially reported by the officer concerned. (Art. 39, M. O. 38.)

Decree of concession.--After the governor general has investigated (in conformance with the provisions in articles 40 to 42, inclusive, of Mining Ordinance 38) an application and the objections thereto, he issues a decree either of refusal or of approval. A decree of refusal must mention the grounds for rejection, and a full copy thereof is given to the applicant, free of charge. (Art. 49, M. O. 38.) A decree of approval shall contain (a) the surname, Christian name, profession, and domicile of the concessionaire, (b) the name of the concession, (c) the area and the boundaries of the concession, marked on a map accompanying the decree, (d) the time for which the concession is granted, (e) the name of the province and the county (or provinces and counties) in which the concession is located, (f) the name of the mineral (or minerals) granted by the concession, (g) the special conditions, if any, and (h) the date of the issuance of the decree. An authentic copy of the decree shall be handed to the concessionaire. (Sec. 2-3, art. 30.) Regulations with respect to the registering and transcribing of a concession or the obtaining of a legal deed of concession are given in articles 100 to 110, inclusive, of Mining Ordinance 38.

The head of the Mining Service may demand in advance payment for the expenses of drawing the maps required for the decree. (Art. 47, M. O. 38.)

The conditions of a concession decree may not be altered or increased in number after the concession has been granted, except with the consent of the holder thereof. (Sec. 2, art. 69, M. O. 38.) The most recent conditions to be imposed are decreed by Statute 1930, No. 348 (October 1, 1930). They have reference to boundary markers; furnishing information with respect to compliance with the mining laws; supplying all data asked for in investigations; presenting by heirs of proof of compliance with article 4 of the Indian Mining Law; reporting changes in the personnel of companies; maintaining production registers; recording and reporting production; furnishing affidavits of the accuracy of reports; recording analyses and assays (if made); appraisals by the Government of taxable commercial products; and similar matters.

Publication.--The provisions for the publication of the application for a concession are practically the same as those governing the application for a prospecting permit. They are covered by article 38 of Mining Ordinance 38. The decree granting a concession is published in full by the Government in the Javasche Courant (art. 48, M. O. 38), as is also the decree rejecting a concession (sec. 2, art. 49, M. O. 38).

Area and boundaries.--A concession is granted for a mining field of indefinite depth, vertically within the limits set forth in the deed of concession. (Sec. 1, art. 13.) An application for a concession may request only one united area forming a coherent unit, the outlines of which are reasonably connected with the discovered mineral deposits. (Art. 33, M. O. 38.) No single concession may be granted for an area larger than 1,000 hectares (art. 35, M. O. 38), except by permission of the governor general and under certain conditions, as indicated in sec. 1, art. 19, and sec. 1, art. 30, of the Indian Mining Law, and article 56 of Mining Ordinance 38. Areas exempted from exploitation are covered under the section of this paper entitled "Exempted Areas".

The corners and boundaries of a concession area must be chosen in conformity with the provisions for prospecting areas, as noted in discussion of areas and boundaries under section entitled "Prospecting." (Art. 34, M. O. 38.)

Provisions with respect to applications for the dividing of concession areas, the interchange of parts of bordering areas, the uniting of concession areas, and like matters (which are possible only by new deeds of concession, according to article 19 of the Indian Mining Law) are covered in articles 111 to 118, inclusive, of Mining Ordinance 38.

Duration.--Mining concessions will be granted by the governor general for a period not to exceed 75 years. (Sec. 1, art. 13.)

In the third year before the date upon which a mining concession terminates, conditions shall be fixed by royal decree whereby a new concession may be obtained, if the Government does not wish either to undertake

the mining itself or to enter into contracts with others for that purpose. The concession holder must make his acceptance of the conditions within six months, or no new concession will be granted except by public competition. (Art. 34.)

Transfer.--The right obtained with a concession is considered immovable property, which may be mortgaged or transferred. (Sec. 1, art. 18.) A permit to transfer a concession may not be transferred (art. 77, M. O. 38) to:-

1. Minors.
2. Persons declared non compos mentis.
3. Limited companies whose activities, according to the statutes, do not include mining operations.
4. Former holders of a forfeited concession (only with respect to the site of the former concession).

The documents of transfer must contain the concession decree in full. (Art. 102, M. O. 38.) Transfer may not be made either to legal heirs or to any others unless those who are to obtain the transfer rights have complied with article 4 of the Mining Law (with respect to citizenship or domicile, etc.). (Sec. 1, art. 103, M. O. 38.) Transfer may not be made until proof has been submitted that the "amounts due as a permanent right" and taxes have been paid. (Sec. 1, art. 45, and sec. 3, art. 103, M. O. 38.)

With the application for a permit to transfer a concession, the copy of the application for the concession held by the concessionaire must be returned to the head of the Mining Service. (Art. 32, M. O. 38.)

Termination of concessions.--Concession rights may be terminated, under certain conditions, by forfeiture, by law, or by request of the concessionaire.

In general, a concession is forfeited if the holder thereof fails to commence work when notified to do so or fails to fulfil the conditions of the concession decree. (Sec. 1, art. 37.) Forfeiture shall not take place until the concessionaire or his representative has been legally served with a written notice that forfeiture is to take place, nor until he has had time to defend his interests, nor until he has been given a year in which to carry out his duties or comply with the demands made. If the failure is with respect to the payment of money, a period shall be prescribed, of at least three months. (Sec. 2, art. 37.) The concessionaire's right of appeal to the Crown, his right to ask for a public sale of the concession for his own benefit, in case creditors have not taken action, and other related matters are covered by article 38 of the Mining Law and by articles 175 to 179 of Mining Ordinance 38. If no sale is requested or if a public auction is without result, the concession is withdrawn. (Sec. 11,

art. 38.) The concessionaire remains responsible for all obligations and shall keep the mining works in proper condition and repair until the transfer to his successor shall have taken place or the concession shall have been withdrawn. (Sec. 12, art. 38.)

A concession is terminated by law, (a) if the owner of the concession or the concessionaire ceases to fulfil the requirements of article 4 of the Mining Law (with respect to citizenship and domicile), (b) at the death of the owner of the right of concession or of the concessionaire, with respect to those successors to his title that have not satisfied the requirements of article 4 of the Mining Law. (Art. 39.)

With respect to requests of the concessionaire for withdrawal of his concession, the same regulations as those mentioned in a preceding paragraph with regard to sale apply. If no request for public auction has been made, the registrar will give the concessionaire a certificate to that effect, which will be forwarded to the Government, with the documents by which it is shown that the request was duly served upon the mortgagees. (See article 40 of the Indian Mining Law.) (See also article 179 of Mining Ordinance 38.)

Section 1, article 41, of the Mining Law, provides that at the termination of a concession all obligations with respect thereto cease, and that the Government obtains full and free rights (with the exception of the parts of the surface and the buildings erected thereon belonging to the last concessionaire) of disposal with respect to the mining field and all that is used for protecting and covering the mining works, no compensation being due to the late concessionaire. Section 2 of the same article decrees that the governor general shall fix the period allowed the concessionaire for the removal of his property. Article 42 of the same law decrees that in case of forfeiture the concessionaire, in the event of the sale of the concession, shall place (free of charge) all charts, drawings, and sketches connected with the mining work at the disposal of the governor general.

When the decision of the governor general cancelling the rights of a concession holder has been served upon him or his representatives, he may not sell or borrow upon the concession further. (Art. 74, M. O. 38.)

Rights and obligations of the concession holder.--In general, the concession gives the concessionaire the exclusive right within the mining field to extract and dispose of all minerals mentioned in the deed of concession and to erect all works necessary thereto, including necessary auxiliary works outside of the concession area. Minerals not mentioned in the deed of concession may also be disposed of, if listed in the mining law; but they require for exploitation a new concession unless their compounds with those mentioned in the grant render their combined extraction unavoidable. (Art. 16.) (See section of this paper entitled "Disposal of Minerals.")

For rights of disposition of the surface within the concession area, see section of this paper entitled "Obligations and Rights of Surface Owners." License to make roads by land or water, as well as the right of superficies to erect the necessary buildings, etc., on State domains, will be granted by the governor general. For ground required for these purposes that is neither within the concession area nor within State domains, the right of expropriation of all properties shall apply. (Art. 22.)

The rights and obligations of one concession holder toward another with respect to the building of works are covered by article 95 of Mining Ordinance 38.

A concession holder is entitled to payment for all damage caused by the building on his concession area of "artificial works," railways, canals, or other public means of transport. (Art. 68, M. O. 38.)

REGULATIONS WITH RESPECT TO PETROLEUM AND RELATED SUBSTANCES

As has been stated in the section of this paper entitled "Rights of Foreigners," the Government of the Netherland Indies has reserved to itself all regions known to contain petroleum. All prospecting and exploiting of petroleum and allied products must be done under contract with the Government, which contracts must be approved by legislative action (art. 5a and 28). A draft agreement, containing detailed regulations, is included in Statute No. 23, the law enacted to authorize the Minister of Colonies to sign four contracts with the Nederlandsche Koloniale Petroleum Maatschappij (Netherlands Colonial Petroleum Co.), a subsidiary of an American oil company, for the "prospecting for and developing of mineral oil, mineral pitch, mineral wax, and all kinds of bituminous substances, both in solid and fluid and in the gaseous state, so far as these do not form part of a fixed rock that for the winning of these substances must be developed in its entirety, consequently iodine and the combinations thereof...."

A surface owner or an interested third party not a holder of a prospecting permit or of a concession (when building works that he has the right to build) upon discovering fluid bitumens and accompanying gaseous substances (on areas not under permit to another nor under contract with the Government) is allowed to dispose (freely and without payment of excise duty) of the freely flowing or streaming substances discovered until necessary measures shall have been taken by the governor general, provided the discoverer gives notice to the head of the local government within eight days after the discovery. (Art. 99, M. O. 38.)

EXEMPTED AREAS

In general, exempted areas include "reserved areas," lands which the Government sets aside for exploitation by itself or by those under contract with the Government (art. 5a), areas to be open to public competition (art. 31 and 32), areas or parts of the country closed by the governor general for

reasons of general interest (subsec. c, sec. 1, art. 8), and "native clearings" (art. 88-93, M. O. 38).

Within the boundaries of prospecting or concession areas, prospecting or mining shall not take place on grounds on which fortifications, government or public buildings have been erected, or churchyards, graves, public roads, canals, or railways have been made, or on grounds considered consecrated according to the religious institutions of the natives, or on grounds on which private dwellings or factories are erected, or within certain distances thereof, unless the holders of surface rights and interested third parties shall have given their consent. (Sec. 2, art. 8, and art. 20.) The specific distances from grounds and buildings are fixed in each case by articles 86 and 87 of Mining Ordinance 38.

DISPOSAL OF MINERALS

Subject to the rights of others and to the provisions concerning rents and royalties, to be discussed subsequently, a prospector is allowed freely to dispose of the minerals that he obtains, in so far as they belong to the first class mentioned in section 1 of the Indian Mining Law (see paragraph (a) under "Minerals Subject to the Indian Mining Laws"). (Sec. 2, art. 10.)

The concession holder, in accordance with the provisions of the law and the conditions placed in the deed of concession, has the exclusive right within his mining field to extract all minerals mentioned in his deed, even in old ore deposits of former operations that may be lying within the boundaries of the mining field. (Sec. 1, art. 16.) The concessionaire has the right to dispose freely (according to the rules laid down by ordinance and for the benefit of his industry exclusively) of such minerals not mentioned in the mining law as he may extract in the working of the mine. (Sec. 2, art. 16.) Minerals coming under the law (except petroleum and allied substances) but not mentioned in the deed of concession may not be worked unless the compounds of those minerals with those mentioned in the grant render their combined extraction unavoidable. (Sec. 3, art. 16.)

For regulations concerning petroleum products discovered by others than the Government or persons or companies under contract with the Government, see section of this paper entitled "Regulations with Respect to Petroleum and Related Substances."

For further regulations concerning minerals for which no right has been granted, especially of minerals discovered during the building of secondary works, see articles 96 to 99, inclusive, of Mining Ordinance 38.

OBLIGATIONS AND RIGHTS OF SURFACE OWNERS

The holders of surface rights and other interested parties must permit prospecting provided (1) that they have been informed beforehand by the license holder of his intention to prospect (the licensee being obligated to produce his license or a copy thereof and to locate the place of the proposed operations) and (2) that an indemnity has been either paid or guaranteed to them, according to regulations to be made by ordinance. (Art. 9.)

The obligation of those having a right to the soil to allow prospecting and mining extends to the building and carrying out of all works and activities required therefor. (Art. 119, M. O. 38.)

If the right of disposition of the surface is required for working purposes for a period not longer than three years, the conditions of article 9, quoted in a preceding paragraph, shall apply to mining also (the provisions with regard to the licensee to be applied to the concessionaire, and the showing of an authentic copy of the document whereby the Government granted the concession to be required). (Sec. 1, art. 21.) If the right of disposition of the surface is required for a period longer than three years, regulations of expropriation for public interest shall apply. (Sec. 2, art. 21.)

Should excavations, digging, surveying, or the placing of poles on another person's land be considered necessary for the marking off of the concession area, those having a right to the land or interested third parties must allow these operations, provided that they have been notified in writing by the head of the local government 48 hours previously (natives or foreign orientals to be notified either in writing or orally by their chief). (Sec. 1, art. 67, M. O. 38.)

The concessionaire is liable for, and must make good to the holders of surface rights, all damage caused by the mining operations, whether they be carried out under the surface or not, or whether the damage has been caused by a deliberate act or could not have been foreseen. (Sec. 1, art. 24.) The details concerning damages to surface owners are covered by sections 2 and 3, article 24, of the Indian Mining Law, by articles 26 and 27 of the same law, and by articles 119 to 150, inclusive, of Mining Ordinance 38, the provisions in the mining ordinance including practice before the courts, both native and European.

RENT AND ROYALTIES

According to article 35 of the Indian Mining Law, the Government levies upon each prospecting permit and upon every discoverer who continues prospecting during the consideration of his application for a concession:

(a) A yearly fixed rent, to be paid in advance each year, proportionate to the area of the prospecting grounds, and at the rate of 0.025 florins[4] ($\frac{1}{2}$d.) a hectare.

(b) A royalty, payable each year, of 4 per cent of the value of the gross yield, after deduction of such quantity of the minerals won as will be declared exempted from taxes by ordinance.

The Government levies on each concession:

(a) A yearly fixed rent, to be paid in advance each year, proportionate to the area of the concession grounds, at the rate of 0.25 florins (5d.) a hectare.

(b) A royalty, payable each year, of 4 per cent of the value of the gross yield.

The calculations and the levying of the royalties (or excise duties) are carried out in the same way for both permit holders and concessionaires, except that in the case of the permit holder or the discoverer, during a calendar year or a part thereof, products up to a value of 5,000 florins may be exempted from tax.

Detailed regulations concerning the calculating and the levying of royalties (or excise duties) are given in article 36 of the Indian Mining Law and in articles 160 to 174, inclusive, of Mining Ordinance 38.

If the concessionaire shall show to the satisfaction of the governor general that the exploitation for the past year was carried on at a loss (or would be if the sums due for the year were paid), the amount of the royalty may be reduced, but not to less than 1 per cent of the gross yield. (Sec. 3, art. 35.) No restitution will be made of moneys duly paid for fixed rent. (Sec. 5, art. 35.)

MISCELLANEOUS

Self-Governing Districts

With respect to the application of the mining law in the self-governing parts of the Netherland Indies, whereof the princes have transferred the right to grant prospecting licenses and concessions to the Government of the Netherland Indies, subsection 2 of section 27 of the Regulation of the Government of the Netherland Indies (Colonial Laws) shall apply. Any exceptions to the mining law for the self-governing divisions of the country shall be made by ordinance. (Art. 44.)

Information to be Kept Confidential

Every officer carrying out the regulations of the Indian Mining Law, of Ordinance 38, or of the regulations issued for their application is

[4] The 1929 exchange value of the florin (or guilder) was 40.1622 cents.

bound to secrecy with respect to everything that he has learned as a result of his official position, except illegal facts, of which he has full authority to notify the officers of the law. Reports of geological mining activities on exploration fields must remain secret until one year after the expiration of the term of the permit, unless the holder of the permit gives written permission for their publication. (Art. 83, M. O. 38.)

Safety Control

Safety control, except in cases specifically exempted by ordinance, is in the hands of the personnel of mine inspection, to whom must be given free admittance to all works and access to all information regarding the observance of the Indian Mining Law, Ordinance 38, and the instructions for their application. Appeals from decisions of mine inspectors may be made to the head of the Mining Service. (Art. 180, 181, and 182, M. O. 38.)

Instructions to be given by Government ordinance concern (a) the solidity of the mining works, (b) protection of the life and the health of the laborers, (c) protection of the surface for the safety of persons and public traffic, and (d) protection from detrimental results of the mining to the public in general. (Art. 183.) These instructions have recently been embodied in the police regulations of mines and mining made effective October 1, 1930, designated as Statute 1930, No. 341.

Fines and Penalties

The fines and other penalties imposed upon prospectors, concessionaires, surface owners, and interested third parties for infringements of the provisions of the mining law and ordinances are covered by articles 184 to 194 of Mining Ordinance 38.

I. C. 6452 February, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

GEOPHYSICAL ABSTRACTS

NO. XXII

BY

FREDERICK W. LEE

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

GEOPHYSICAL ABSTRACTS[1]

No. 22

Compiled by Frederick W. Lee[2]

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6452."

2 - Senior physicist, U. S. Bureau of Mines.

1. GRAVITATIONAL METHODS

(42) A REDETERMINATION OF THE CONSTANT OF GRAVITATION

By Paul R. Heyl

Bureau of Standards Journal of Research, vol. 5, No. 6, 1930, pp. 1243-1291.

Contents of the article:

Author's abstract reads as follows: "A redetermination of the constand of gravitation has been made, using the torsion balance in vacuum. The large masses were steel cylinders weighing about 66 kilograms, and the small masses spheres of gold, platinum, and optical glass. The final result obtained is 6.670 x 10^{-8} cm^3 g^{-1} $sec.^{-2}$.

"While the results obtained appeared to differ slightly with small balls of gold, platinum, or glass, it was shown by a special experiment with the form of torsion balance used by Eötvös that this is not to be attributed to difference in material."

Fourteen figures and 20 tables illustrate the article.--W. Ayvazoglou.

(43) GRAVIMETRICAL OBSERVATIONS OF THE BEREKEY OIL REGION DURING THE YEARS 1927 AND 1928 (IN RUSSIAN)

By S. P. Poletajev

Neftianoe Khoziaystvo, vol. 19, No. 10, 1930, pp. 385-392.

The problem of the gravimetrical survey of the Berekey oil region consisted of investigating the anticline fold and establishing the structure of the dome of the foraminiferous layers at the top of which a petroleum horizon was known. This question could not be solved satisfactorily by geologists, owing to the absence of outcrops and insufficient correctness in old geological maps.

The data obtained by gravimetrical survey did not agree with the old representation of the geological structure of the region of Berekey and the direction of the axis of the anticline was found to be different from that expected by geologists. The results of the gravimetrical survey are given in two maps, a series of crosscuts and schemes showing the structure of the Berekey dome as based on the data obtained during 1927 and 1928.

The following conclusions were drawn:

1. It was proved by the gravimetrical work that the geological structure of the region of Berekey was more complicated than could be expected on the basis of the data obtained from drilling.

2. There is a break of foraminiferous layers along the anticline axis not only in the territory of the Berekey petroleum field but also to the north of it.

3. The foraminiferous layers in the region of the oil fields are not distributed in accord with chalk and probably are separated from it by less solid masses (clays).

4. In order to explain the inconstancy in the difference of the mean densities of masses situated above and below the top of the foraminiferous layers, it must be assumed that the foraminiferous layers in the region of Berekey are divided into thin scales.--W. Ayvazoglou.

(44) ELEMENTS OF ISOSTASY -- OBSERVATIONS AND INTERPRETATION

By William Bowie

Scientific Monthly, August, 1930, pp. 163-176.

Taking into consideration the importance of isostasy as a branch of geophysical science, Bowie describes in this article its elements, observations, and interpretation under the following headings: 1. Measuring of isostasy; 2. Isostasy in India (J. Pratt and G. Airy); 3. Deflections from the vertical and their causes; 4. Isostasy in the United States; 5. Maintenance of isostatic equilibrium; 6. The earth a yielding mass; 7. Study of gravity anomalies by grouping stations; 8. Explanation of gravity anomaly;

9. Studies of isostasy in Canada; 10. Determination of depth of compensation; 11. Horizontal extent of compensation; 12. denudation and sedimentation; 13. Mountain formation; 14. Relation of gravity anomalies to geological structure.

The author concludes that much more could be said in regard to the importance of keeping isostasy constantly in mind when one tries to solve the great structural and dynamic problems of geology; and that undoubtedly the literature of the future will treat isostasy at even greater length than has been the case in the past.--W. Ayvazoglou.

(45) ISOSTATIC COMPENSATION IN RELATION TO GEOLOGICAL PROBLEMS

By George R. Putnam

The Journal of Geology, vol. 38, No. 7, 1930, pp. 590-600.

The bearing of gravity results on general geological problems is made more clear by recognizing the fact that complete local isostasy is impossible, and that such an assumption leads to appreciable errors in gravity anomalies in mountainous regions. The horizontal and vertical distributions of isostatic compensation are interrelated, and should be studied together. The ranges of assumptions as to both are stated, with the probable limits. The truth of regional instead of local isostasy has important bearings on geological deductions, and makes the gravity results more consistent with other geological evidence as to the earth's crust.--Author's abstract.

2. MAGNETIC METHODS

(46) DIE MAGNETISCHEN EIGENSCHAFTEN DER ERUPTIVGESTEINE

(MAGNETIC PROPERTIES OF ERUPTIVE ROCKS)

By Kurt Puzicha

Zeitschrift fuer praktische Geologie, vol. 38, No. 11, 1930, pp. 161-172, and No. 12, 1930, pp. 184-189.

Contents of the article:

1. Introduction.

2. Critical consideration of methods used so far, as well as of the results.

3. Stating of the problem.

4. Arrangement of the experiments: (a) theory; (b) constructive data; (c) adjustment.

5. The results: (a) general; tabular classification of the results of measurements; (b) the dependence of the susceptibility on the field strength; (c) the dependence of the susceptibility on the mineral components of the rocks; (d) the dependence of the coercive force and of the remanence on the mineral components of the rocks.

The author's summary of the article reads as follows:

With the aid of a sensitive ballistic measurement process the general magnetic properties of eruptive rocks were examined on a large number of samples.

It was established that:

1. The susceptibility was independent of the field strength only in a small part of the rocks examined.

2. The susceptibility decreased with the increase of the field strength only in exceptional cases; in most cases it increased with the increase of the field strength. This increase reached about 230 Gauss, almost 180 per cent (65 per cent on the average).

3. The character of the susceptibility depended mainly on the content of magnetite; the increase was proportionally smaller in magnetic pyrites. A conclusion on the amount of susceptibility could not be drawn from the quantitative determination of the content of magnetite only; the size of grains was to be taken into consideration also.

4. The grade of magnetizability did not depend on the content of titanium in the magnetite; ilmenite did not prove to be magnetizable.

5. The coercive force amounted to several hundred Gauss. It depended mainly on the content of iron oxides, magnetic pyrites, and titanium-free magnetite. Titanium magnetite possessed high susceptibility but only limited coercive force.--Author's abstract translated by W. Ayvazoglou.

(47) TERRESTRIAL MAGNETISM FROM THE VIEWPOINT OF THE ENGINEER

By N. H. Heck

Scientific Monthly, April, 1930, pp. 326-341.

A description of instruments and proceedings for obtaining data on magnetic surveying, illustrated by photographs and diagrams, is given.

The difficulties of studying all the different phenomena associated with terrestrial magnetism, as well as the difficulty of completely understanding them and predicting future conditions with accuracy, are mentioned.

The author predicts, that just as in many other fields of physics the investigations started for a definite purpose have produced wholly unanticipated results, the engineers interested in terrestrial magnetism, although not accustomed to undertake problems for which the complete solution is far ahead, as engineering research is established on the basis of immediate or early financial return, will finally also have to engage in research which has no immediate prospect of financial return.--W. Ayvazoglou.

(48) SUR LE LEVÉ MAGNÉTIQUE DES KARPATES DE SKOLE ET DE LEUR AVANT-PAYS

(ON THE MAGNETIC SURVEY IN THE SKOLE-CARPATHIANS AND THEIR FORELAND)

(IN POLISH)

By E. Stenz and H. Orkisz

Compte Rendu du 1-er Congrès de la géologie du pétrole à Lwow, 14-15 December, 1929, pp. 97-103.

Two magnetic surveys were carried out in the Skole-Carpathians during 1929: one absolute and one relative. The absolute survey has been carried out on an area of about 4,050 square kilometers within the limits of the 1 : 75,000 maps of Drohobycz, Skole, Bolechów and Zydaczów.

Two dip-needles manufactured by Chasselon in Paris and one magnetic theodolite (small model) were used for the measurements. The distance between the points of magnetic inclination was 4 kilometers, that between the points of the horizontal component 6 kilometers, and between the points of declination 6 kilometers.

In general there were made: 248 measurements of inclination, 65 measurements of the horizontal component, and 35 measurements of declination. All the results were reduced to the same epoch: 1928.5.

The relative survey has been carried out for the purpose of prospection. It consisted of 990 points of measurements of the vertical component carried out within the limits of the polygon Stryj-Stebnik-Synowódzko-Bolechów-Zurawno-Zydaczow-Stryj. The density of the points was equal to 1 kilometer. Schmidt's magnetic vertical balances were used for the observations.

Records obtained with the Askania-Werke magnetograph, installed in Daszawa, near Stryj, were used for the reduction of the results of measurements.

Maps showing (1) the horizontal component of the magnetic field, (2) the inclination, and (3) the anomalies of the inclination are added to the article.

The magnetic survey shows an interesting agreement between the course of the isodynamics and the direction of the Carpathian "skibas" (blocks driven one upon the other in the direction, in this region, from southwest to northeast).--Author's abstract translated by W. Ayvazoglou.

(49) RÈGLES PRATIQUES POUR L'EMPLOI DU MAGNÉTOMÈTRE DANS LES PROSPECTIONS GÉOPHYSIQUES

(PRACTICAL RULES FOR THE USE OF THE MAGNETOMETER IN GEOPHYSICAL PROSPECTING)

By M. C. Alexanian

Annales de l'office national des combustibles liquides, vol. 5, No. 4, 1930, pp. 677-702.

Contents of the article:

Introduction.

Sources of error arising from: (1) construction; (2) observation; (3) natural causes.

I. Errors arising from construction: (1) Errors in construction; (2) displacement of the center of gravity of the magnetic system; (3) displacement of the zero point of the scale; (4) determination of the constants. Scale-constants (vertical component, horizontal component); (5) temperature constants; (6) variation of the horizontality of the apparatus; (7) conditions of the knife-edges.

II. Errors arising from observation: (1) Choice of stations; (2) setting up of the instrument; (3) position of the operator; (4) carrying out of measurements; (5) reading of the scale; (6) determination of the east-west magnetic direction. Influence of the bad orientation of the instrument; (7) final calculations.

III. Errors due to natural causes: (1) Displacement of geographic coordinates; (2) daily variations; (3) magnetic storms; (4) electric currents.

IV. Criticism of the results of measurements: (1) Accuracy of measurements; (2) agreement between the points of measurement; (3) magnetic profiles; (4) verification profiles; magnetic maps; (5) representation of numerical operations; (6) number of stations which can be completed during one day.

The different sources of error which may enter into the course of operations during a magnetic survey, as enumerated in the contents above, are discussed.--W. Ayvazoglou.

3. SEISMIC METHODS

(50) EARTHQUAKES, A CHALLENGE TO SCIENCE

By N. H. Heck

Scientific Monthly, August, 1930, pp. 113-126.

After a brief description of a few different types of earthquakes, which may serve for the purpose of illustration, the author discusses the

challenges to science presented by them, the first of them being the cause of earthquakes. Concerning the relation of volcanoes to earthquakes, the author says that the volcano seemingly is a localized and rather superficial phenomenon as compared to the earthquakes, that generally earthquakes are due to the slipping of two rock surfaces on one another, and that it is peculiarly helpful to study the effect at the surface. Some examples supporting this supposition are given.

The necessity for accurate determination of the positions and elevations of fixed points in order to study the surface effects caused by earthquakes is mentioned.

The method of attacking the problem of studying the point of origin of an earthquake by measuring the time of travel of waves produced by large explosions, since in this case the point of origin is known, has resulted in the development and confirmation of the important earthquake wave theory. Principal earthquake waves are discussed and their paths shown in figures. The theory of isostasy is mentioned and illustrated by a figure. Two maps, one showing the places of known earthquakes of the United States and another the distribution of seismological stations of the United States and adjacent Canada are added.

The last part of the article deals with instruments for recording earthquakes. Photographs of the Wood-Anderson torsion seismometer and the Wenner seismometer, as well as the record produced by the latter, are given.-- W. Ayvazoglou.

(51) ON THE RELATION BETWEEN THE SUNSPOT NUMBER AND THE DESTRUCTIVE EARTHQUAKES IN JAPAN (IN JAPANESE)

By Takeo Takayama and Takeo Suzuki

Bulletin of the earthquake research institute, Tokyo Imperial University, vol. 8, No. 3, 1930, pp. 364-374.

The present investigation is a statistical study on the relation between sunspot activity and the destructive earthquakes in Japan (Formosa excluded) for 318 years from 1608 to 1925. (Time and space distribution of destructive earthquakes is shown in a figure.) The method of this investigation is that the number of destructive earthquakes which occurred for three years containing the year of sunspot maximum or minimum and compared to the same in the other years, have been counted and reduced to the frequency in an equal duration, which at least were represented by the percentage frequency. Similarly the earthquakes percentage for five years of sunspot maximum or minimum was calculated.

The values of percentage given in four tables in the text are the mean values of these two cases. The authors took statistics dividing the whole time interval into two or four and also the space into three zones, i.e., Omori's inner (Japan Sea side) and external (Pacific side) seismic zones and the inland zone of earthquakes which is the aggregate of several local seismic zones between the above two. The result obtained is summarized as follows:

1. The authors could not find any noteworthy relation from the statistical data taken over the whole Japanese area (Table 1).

2. When, however, the three seismic zones are considered separately, there exist noteworthy relations, that is: (1) In the inner seismic zone the earthquakes occurs more frequently in the vicinity of the sunspot maximum; (2) in the external seismic zone, on the contrary, the earthquakes occurs more frequently in the vicinity of the sunspot minimum; and (3) in the inland or middle seismic zone the relation is not noteworthy (the relations are shown in figures). These facts suggest some possible relation in connection with the shift of the zone of the atmospheric high pressure.

3. On the cyclicity of the destructive earthquakes in Japan for from 660 to 1925, the curves of fluctuation of the seismic frequency have been constructed by taking various time intervals as unit of time in which the frequency is counted.

Each curve showed the apparent cyclic nature with its own period, which is found to be three or four times as long as the assumed unit interval as shown in a table. This clearly shows that the apparent periodicity is not the real one but is subjected to the accidental phenomena completely investigated by Prof. T. Terada.--Author's abstract.

(52) RESULTS OF ELASTIC-WAVE SURVEYS IN CALIFORNIA AND ELSEWHERE

By Frank Rieber

Bulletin of the American Association of Petroleum Geologists, vol. 14, No. 12, 1930, pp. 1557-1571.

The author's abstract reads as follows:

"Geophysical exploration in loosely consolidated materials presents a problem different from similar surveys in areas where there is a marked differentiation between beds. However, even in these loosely consolidated materials, velocity of wave transmission varies in general with depth of original overburden, and correlations based on this fact have been found reliable. Apparatus has been specially developed to meet the requirements of operation in such materials."

The question is discussed under the following headings:

1. Relation of lithology to velocity characteristics;

2. Relation of velocity to depth of original overburden;

3. Accuracy requirements in mapping loosely consolidated sediments;

4. Special features of interpretation of results;

5. Requirements for apparatus for mapping loosely consolidated materials.

Sixteen figures illustrate the article.--V. Ayvazoglou.

(53) THE SCREENING OF SOUTHEND FROM GUNFIRE

Editorial note

Engineering, vol. 130, No. 3387, 1930, pp. 731-732.

The mechanism by which velocity and temperature gradients cause the refraction of sound is discussed. The basis of refraction is, briefly, that the speed of sound propagation is increased by use of air temperature and increase of wind speed. If a sound wave-front be imagined to impinge obliquely on a stratum of air in which the temperature rises with altitude, the upper part of the wave, which penetrates first into the higher-temperature region, travels faster then the lower part, and the inclination of that portion of the wave to the horizontal is increased. If the instantaneous direction of propagation is normal to the wave-front, the sound is gradually refracted toward the earth. Similarly, if the wind component in the direction of the sound increases with height, the waves are refracted downwards. Conversely, if the wind and temperature gradients be such as to reduce the speed of sound as altitude increases, the wave-front is refracted upwards. These effects are illustrated by diagrams. A formula for the height of the acoustical shadow for a certain distance from the source of sound is derived.

An example of the application of the existing approximate methods concerning acoustic investigations are given by Dr. W. S. Tucker in connection with the screening of the north side of the Thames Estuary from the noise of 16-inch gunfire practice in the Isle of Grain.

Experiments with seismographs showed that no appreciable disturbance was transmitted through the ground. Therefore investigations of the "onde de choc" (sound due to the compression wave at the nose of a high-velocity projectile) were carried out. For the prediction of suitable firing conditions photographic records of gun reports were taken by sound ranging apparatus at three stations along the Southend front. The records are reproduced in figures.

In order to facilitate the forecasting of safe firing conditions, Tucker devised a series of celluloid disks, shaped like the shadow diagrams shown in three figures. To ascertain meteorological conditions, the approximate wind gradient disk is placed on a map of the district, with its center at the gun, and oriented to the wind direction prevailing at the time. The approximate temperature gradient disk is superimposed, and the height of the resultant shadow at the range (5 miles) for which the disks are designed may be read off to scale.--W. Ayvazoglou.

4. ELECTRICAL METHODS

(54) ELECTRICAL PROSPECTING

By Allen A. Rogers

The Tech Engineering News, vol. 11, No. 6, 1930, pp. 218-219 and 243.

After a brief mention of geophysics as an agency of prospecting for minerals and oil, the author describes the application of electricity for

geophysical prospecting. The method of equipotential lines and that of induction are discussed. A map of current and equipotential lines and a plan and profile views of an ore deposit located by electrical prospecting are given. In conclusion Rogers points out that geophysical prospecting, being a new tool for the geologist and mining engineer, can be used only by those trained in this art. As an aid to extending the scope of the results of geological studies, to finding and tracing certain concealed features of the earth's crust, it is invaluable and its use is bound to increase.--W. Ayvazoglou.

(55) MESSUNG DER ELEKTRISCHEN ERDBODENEIGENSCHAFTEN ZWISCHEN 20 UND 2.10^7 HERTZ. (CYCLES PER SECOND)

(MEASUREMENT OF ELECTRICAL PROPERTIES OF THE SOIL BETWEEN 20 AND 2.10^7 HERTZ)

By M. J. O. Strutt

Elektrische Nachrichten-Technik, vol. 7, No. 10, 1930, pp. 1-8.

Contents of the article:

1. Importance of the electrical properties of the soil.

2. Measurements between 20 and 6.10^5 Hertz.

3. Arrangements for measurements between 6.10^5 and 2.10^7 Hertz.

4. Calibration of the tubular voltmeter in case of short waves.

5. Results of measurements in case of short waves.

6. Discussion of the results.

7. Summary.

The author measured the electrical conductivity between 20 and 5.10^5 Hertz, as well as the electrical conductivity and the dielectric constant between 6.10^5 and 2.10^7 Hertz (undamped) in a moderately moist meadow ground. The following results were obtained:

1. The conductivity increased between 20 and 500 Hertz to about 30 per cent (polarization) and then remained practically constant up to 2.10^7 Hertz.

2. This frequency course remained approximately valid also in case of a slight change of the water content in the soil.

3. Contrary to the author's expectation, after a rain the damping influence of the soil upon the electrical wire waves decreased for waves below 200 meters.

4. The dielectric constant increased, owing to the rain, relatively more than the conductivity -- this explained the conditions mentioned in No. 3.

5. The dielectric constants were about 10 and 15 (on the same ground after rain). Between 10^6 and 2.10^7 Hertz they depended on frequency to a slight degree only.--Author's abstract translated by W. Ayvazoglou.

(56) ELECTRICAL PROSPECTING IN THE REGION OF GROZNY (IN RUSSIAN)

By D. Golubiatnikov

Azerbeidjanskoe Neftianoe Khoziaystvo, vol. 10, No. 10, 1930, pp. 51-54.

The author describes the application of Schlumberger's method for investigation of rocks along the drilled holes ("electrical logs") in the region of Grozny. Field work carried out by Schlumberger's method of electrical prospecting is discussed. The methods are briefly described. The results are summed up as follows:

Investigation of rocks along the drilled holes by the method of electrical logs (carrotage) proved to be very satisfactory; several oil-bearing and water-bearing horizons not mentioned in the geological structural maps were established. Schlumberger's method of electrical prospecting in places covered with alluvium where good results can not be secured by digging is the sole means by which a quick answer on the conditions of the structure of the subsoil can be obtained.--W. Ayvazoglou.

(57) ELECTRICAL PROSPECTING IN THE REGION OF GROZNY (IN RUSSIAN)

By D. Golubiatnikov

Za Neftianuiu Piatiletku (for the 5-year oil industry program), No. 11-12, 1930, pp. 11-14.

The author describes the electrical prospecting work carried out according to Schlumberger's method in the region of Grozny. Six parties were intrusted with the examination of rocks along the holes, investigation of sections of the holes, establishment of oil-bearing and water-bearing horizons, determination of the structural details of the Novo-Grozny and Staro-Grozny regions, search for new structures in the area adjoining the industrial regions and finally with the establishment of the possibility of finding the raising of the anticline toward the great plain to the north of the industrial region. All these geological tasks were solved by electrical prospecting very successfully.--W. Ayvazoglou.

5. RADIOACTIVE METHODS

(58) KENNELLY-HEAVISIDE LAYER HEIGHT OBSERVATIONS FOR 4,045 AND 8,650 KC.

By T. R. Gilliland

Bureau of Standards Journal of Research, vol. 5, No. 5, 1930, pp. 1057-1068.

Virtual heights of the Kennelly-Heaviside layer as measured by the radio-echo method are reported for 4,045 and 8,650 kc. The report covers daytime observations made each week between January 16 and June 19, 1930. Two

evening tests are also described. Curves are given comparing heights with sunspot numbers and magnetic character. Records taken on April 28, 1930, the day of the solar eclipse, are shown.--Author's abstract.

(59) RADIOACTIVE PROPERTIES OF ROCKS, SOILS, CRUDE OIL AND WATERS FROM SOUTHERN CALIFORNIA

By J. Lloyd Bohn

Journal of the Franklin Institute, vol. 210, No. 4, 1930, pp. 461-472.

This paper includes measurements on waters from Lake Arrowhead, Arrowhead Hot Springs, Harlem Hot Springs, the Pacific Ocean, and from numerous wells and tunnels of the Pasadena and neighboring water supplies. It also includes measurements on rocks, soils, and crude oil.

The activity of the water of Lake Arrowhead is found to be one hundred times smaller than the minimum that could be detected with the electroscopes used for cosmic ray measurements by Millikan and Cameron. This lake is, therefore, very suitable for cosmic ray measurements. The Pasadena wells yield waters approximately one thousand times as active as the Lake Arrowhead water, thus making the Pasadena reservoirs unsuitable for these measurements.

The ratio between the activity of the soil at Lake Arrowhead and that of the soil from the Institute campus, as obtained by Millikan and Cameron from their cosmic ray data, is in good agreement with the results in this report. However, in this report no account is taken of the γ-radiation due to the thorium series, which is of the order of, and probably slightly exceeding, that due to the radium series. If the Th|Ra ratio is constant, as is usually the case, then the results of the two methods are in good agreement.

The average radium content of the acid intrusive rocks in the vicinity of Devil's Gate Dam, Pasadena, is found to be 2.3×10^{-12} grams of radium per gram of rock.

Pasadena tap water contains about 1 eman or 1×10^{-10} curies per liter, while the average of the sources is about five times as great. The radium emanation found in crude oil was of the same order of magnitude as that found in well water.

Carnotite when fused with carbonates at about 1,000° C. loses part of its Ra — probably as a sublimate of $RaBr_2$.

Such a loss in fusing rocks might account for the discrepancy between the solution and fusion methods as found by J. Joly.--Author's abstract.

(60) UNTERSUCHUNG UEBER DIE BEZIEHUNG DER HÖHENSTRAHLUNG ZU ERDMAGNETISCHEN STÖRUNGEN

(INVESTIGATION ON THE RELATION OF PENETRATING RADIATION TO EARTH-MAGNETIC DISTURBANCES)

By Walter M. H. Schulze

Physikalische Zeitschrift, vol. 31, No. 22, 1930, pp. 1022-1025.

As long as the penetrating radiation was considered to be a very hard γ-radiation no relation between the intensity of radiation and the earth-magnetic phenomena was expected theoretically; therefore no experimental investigation has been made in this line. This question became of practical interest after the results obtained by investigations carried out by Bothe and Kohlhörster with Geiger's electron counter, according to which there were found, in additionto γ-rays, also rays of very high penetration, that is very rapid corpuscular rays, probably of the type of β-rays.

In this article the author gives the results of his studies, together with those of Corlin, concerning the exact calculation of correlation between the intensity of penetrating radiation and single earth-magnetic elements.

At the disposal of the author was a series of penetrating radiation-intensity measurements carried out by Corlin, as well as observations on the most important meteorological, magnetic, and aurora borealis phenomena in Abisco made from October 16, 1929, to January 12, 1930.

The measurements were made with Büttner's radiation apparatus A of Kohlhörster's type III. The results of the measurements are given in a series of tables.

A noticeable influence of magnetic declination disturbances could not be established on the basis of the investigations made. The question of the effect of the changes of the horizontal and vertical intensity upon ionization will be taken up for study.

In conclusion the author mentions briefly the relations to aurora borealis as established by investigations made by him.--W. Ayvazoglou.

(61) SCATTERING OF HARD γ-RAYS

By C. Y. Chao

The Physical Review, vol. 36, No. 10, 1930, pp. 1519-1523.

In a previous study of the absorption coefficient of hard γ-rays in various elements (Chao, Proc. Nat. Acad. Sci., vol. 16, 1930, p. 431) the author has found that the absorption coefficient of light elements was predicted fairly well by the Klein-Nishina formula, which assumes that the removal of the energy from the primary beam is entirely due to Compton scattering of the extranuclear electrons. For heavy elements, however, the experimental

value was much larger than was to be expected from the Klein-Nishina formula or any other. Two causes can be suggested to explain this additional absorption: (a) It may be an extranuclear phenomenon due either to an ordinary photoelectric absorption or a breakdown of the Klein-Nishina formula for Compton scattering in these elements; (b) it may also be a nuclear phenomenon, such as the scattering by particles inside the nucleus or any other nuclear absorption (like the excitation of the photoelectric effect occurring there). In an attempt to obtain more information about these questions, a study of the scattered rays has been made.

The results obtained from this study are summed up by the author as follows:

Measurements have been made on the scattering of γ-rays from Th C by Al and Pb. For Al the scattering is, within experimental error, that predicted by the Klein-Nishina formula. For Pb additional scattered rays were observed. The wave-length and space distribution of these are inconsistent with an extranuclear scatterer, and hence they must have their origin in the nuclei.--Author's abstract.

(62) ON THE QUESTION OF THE CONSTANCY OF THE COSMIC RADIATION AND THE RELATION OF THESE RAYS TO METEOROLOGY

By Robert A. Millikan

The Physical Review, vol. 36, No. 11, 1930, pp. 1595-1603.

Mean cosmic-ray intensities have been measured with much precision both at Pasadena, Calif. (latitude 34), and at Churchill, Manitoba (latitude 59), the latter a distance of 730 miles from the North magnetic pole.

1. The observed equality in these intensities indicates that these rays enter the earth's atmosphere as photons rather than as streams of electrons.

2. Evidence is presented that the incoming rays are of a uniform intensity in all directions and in all latitudes, the small and apparently erratic fluctuations found by many observers at different stations arising simply from eruptions, waves, or ripples which change the thickness of the atmospheric blanket interposed between the source and the observer.

3. The cosmic-ray electroscope thus acquires significance as a meterological instrument.

4. The influence of these rays in the maintenance of the earth's charge is considered.--Author's abstract.

(63) THE SIGNIFICANCE OF RECENT COSMIC-RAY EXPERIMENTS

By R. A. Millikan and I. S. Bowen

Proceedings of the National Academy of Sciences, vol. 16, No. 6, 1930, pp. 421-425.

The particular recent cosmic-ray experiments, the significance of which is discussed in this article, are (1) as yet unpublished results of work by Millikan and Cameron on absorption coefficients of cosmic rays on high mountains and at great depths in mountain lakes, (2) recent experiments by Millikan and Bowen on the absorption of gamma rays in mountain lakes, (3) new experiments in the Norman Bridge Laboratory by Chao on gamma-ray absorption, and (4) experiments by Bothe and Kohlhörster and by Curtiss on coincidences obtained with the use of cosmic rays in Geiger-Müller ionization counters.

The authors revealed, in accord with Regener's finding, a very weak, very penetrating radiation at great depth beneath the surface of Gem Lake, 300 feet and more, due, according to the atom-building hypothesis, to the formation of the heavy and rarer elements out of hydrogen. They also revealed at great altitudes on mountain peaks the steeper ionization-depth curve predicted by the Klein-Nishina formula from the hypothetical formation of helium out of hydrogen.

The penetrating power of the softest of the cosmic rays was found to be roughly five times that of these gamma rays of Th C^{11}. This relative penetrating power corresponds, according to the Klein-Nishina formula, to an energy of the softest cosmis rays about ten times that of these hardest gamma rays, and this is also the relative energy of the gamma rays from Th C^{11} and the energy of the softest cosmic ray as computed from the Einstein equation, the Aston curve and the assumption that the softest cosmic ray is produced by the formation of helium out of hydrogen. These facts support strongly the atom-building theory.

The experiments are of great importance because they show that beta rays of the enormous energies involved in the cosmic rays have a penetrating power of the same order of magnitude as the cosmic rays themselves. This is a new and an important discovery, and it is the whole significance of these experiments.

Thus if energies of considerably more than 500 million volt-electrons could be established, then the theory of atom building in interstellar space as the source of the cosmic rays would have to be abandoned.--W. Ayvazoglou.

7. UNCLASSIFIED METHODS

(64) GEOPHYSICAL NOTES ON CALIFORNIA AREA

By Paul B. Whitney

The Oil and Gas Journal, vol. 29, No. 32, 1930, pp. 32, 146-153.

The author expresses his confidence that geophysical methods will be extensively employed in California for solution of unknown structural

conditions, particularly in the San Joaquin and Sacramento Valleys, as nearly all of the geophysical methods (electrical, electromagnetic, torsion balance, seismograph, and magnetic) have given practical results in California.

Some results of geophysical surveys carried out in California are explained, based on curves, graphs, isograms, and profiles given in the article.

The outlook for the future application of geophysical methods in California is especially promising, taking into consideration that the cost of establishing and maintaining an efficient geophysical department is less per year than the cost of one dry hole, thus, with expectation of but one new discovery in two years time, the maintenance of a strong geophysical department is economically more than justified for any oil company.--W. Ayvazoglou.

(65) GEOPHYSICAL PROSPECTING IN 1930

By Donald H. McLaughlin

Mining and Metallurgy, vol. 12, No. 289, 1931, pp. 22-26.

The increasing use of geophysical methods by geologists and engineers during the past year is mentioned as a significant and an encouraging sign.

The relation of geophysical and geological work has become increasingly clear.

A slump in geophysical exploration was noticeable in the Gulf Coast region during the past six months on account of the contrast to the great activity at the beginning of the year. The subsequent sudden decrease in geophysical prospecting in this region can probably be attributed chiefly to the depression in the oil business, rather than to discouragement over results, but to an important extent it was due to the completion of surveys in the most promising areas.

The efforts to secure records from greater depths with the seismic methods have brought about many revisions leading to greater accuracy in time-distance observations. Commercial magnetometer surveys have been greatly restricted during the past six months, but the usual steady accumulation of records from scientific work in this field still continues.

Electrical prospecting during the year continued to be applied in a wide range of geological problems, from structural work in petroleum fields to foundation investigations for dams. In the search for metallic ores, satisfactory results continue to be reported from districts where conditions are suitable for effective work.

In conclusion the author mentions the following three meetings held during the year that were of particular importance with respect to presentation of papers on geophysical prospecting and for the exchange of information: (1) The American Institute of Mining and Metallurgical Engineers, (2) the American

Association of Petroleum Geologists, and (3) the Fourth General Assembly of the International Union of Geodesy and Geophysics held in Stockholm, Sweden.

The widening interest in geophysical prospecting in England, the publication of the special supplement to Gerland's Beiträge zur Geophysik, entitled Ergänzungshefte für Angewandte Geophysik, the appearance of the Boletin de la Asociacion Geofisica de Mexico, as well as the work of the U. S. Bureau of Mines, are mentioned.--W. Ayvazoglou.

(66) GEOPHYSICAL PROSPECTING POSSIBILITIES FOR TIN IN THE FEDERATED MALAY STATES

Editorial note

The Mining Journal, London, vol. 170, No. 495b, 1930, pp. 658-659.

This is a portion of the report on the application of geophysical methods of prospecting for the investigation of tin ores. The report was made by Mr. Broughton Edge based on his visit in the Federated Malay States from July 22 to August 2, 1929. It embodies the conclusions arrived at after discussion with the Director and other members of the F. M. S. Geological Survey Department, together with visits to the principal tin producing centers in Perak and Selangor.

The following kinds of tin deposits were investigated:

1. Alluvial and residual tin deposits.

2. Sulphide bearing tin-pipes in limestone.

3. Sulphide bearing tin lodes in granite.

4. Sulphide bearing tin lodes in schist.

The conclusions drawn read as follows:

Geophysical methods of prospecting, in their present stage of development, can not be applied extensively to the mining fields in Perak and Selangor which were visited -- or to those in other parts of the F. M. S. where the modes of occurrence are similar.

In the cases of alluvial and eluvial tin deposits there is little prospecting of success by any geophysical method, but the sulphide bearing tin lodes and pipe deposits occurring in granite, schist, and limestone present more favorable conditions. There is good reason to believe that certain of the geoelectrical methods would be effective in locating them.

In particular, important possibilities are presented by the sulphide bearing tin-pipes which are characteristically developed in the limestone country along the eastern flank of the Kledang Range (Kinta) and if arrangements could be made for a geoelectrical examination of

the ground which extends between Lahat and Selibin (Kinta district) there would be good prospects of success.

In some of the tin fields it is possible that occasions may arise in which data are required regarding the depth and configuration of the limestone surface over large areas, in which the depth of alluvium is too great to permit a thorough boring compaign (e.g. 200 feet or more). Under such circumstances it is likely that a seismic survey would prove effective.

In the Batu Arang coal field it is probable that some useful work could be carried out by the gravimetric method. This would be limited, however, since the topography over the great part of the field is too rough to permit a general application of the system.--W. Ayvazoglou.

(67) LES HOMOLOGIES PODOLIENS-KARPATIQUES; LEUR APPLICATION AUX RECHERCHES GÉOPHYSIQUES DANS LA ZONE SUBKARPATIQUE

(PODOLIA-CARPATHIAN HOMOLOGIES; THEIR APPLICATION TO GEOPHYSICAL INVESTIGATION IN THE SOUTH CARPATHIAN ZONE) (IN POLISH)

By W. Teisseyre

Compte Rendu du 1-er Congrès de la Géologie du Pétrole à Lwow, 14-15 Dec., 1929, pp. 37-65.

After a detailed geological description of the region the author mentions the directions for geophysical investigations based on a series of displacements and warpings in Podolia which, he states, agree with the details in the changes established by recent surveys. The positions of culminations and depressions with regard to the dislocations in the foreland will become valuable by the fact that, after having cancelled the illusory culminations, the comparison of the intervals between the main culminations with those between the great structural lines of the foreland can be established.--W. Ayvazoglou.

(68) DE L'APPLICATION DES MÉTHODES GÉOPHYSIQUES AUX RECHERCHES DE LA GÉOLOGIE DU PÉTROLE DANSLES KARPATES ET L'AVANT-PAYS

(ON THE APPLICATION OF GEOPHYSICAL METHODS OF PROSPECTING GEOLOGICAL STRUCTURE OF PETROLEUM FIELDS IN THE CARPATHIAN MOUNTAINS AND THE FORELAND (IN POLISH)

By E. W. Janczewski

Compte Rendu du 1-er Congrès de la Géologie du Pètrole à Lwow, 14-15 Dec., 1929, pp. 81-97.

The author reviews the principal methods of geophysical prospecting (gravimetrical, magnetic, radioactive, geothermic, seismic, electrical) and characterizes the kinds of indications which can be furnished by these methods on the nature of the subsoil.

He discusses the use of the Sterneck pendulum for the study of the distribution of the gravity anomalies in the Carpathian Mountains and the foreland and believes that these investigations will contribute to the determination of the main lines of deep tectonics of these regions, as well as their relationship to the Podolian plateau.

In speaking of the advantages established by the application of the method of artificial seismic oscillations, he emphasizes the importance of combining it with the gravimetrical method in order to obtain a more exact representation of the subterranean structure of the foreland of the Carpathian Mountains.

The other methods are discussed with reservation, owing to the local conditions of the peripheral depression filled up by a series of monotonous Tertiary layers in which very saline underground waters are circulating.--Author's abstract translated by W. Ayvazoglou.

(69) TRAVEAUX EN TERRAIN EXECUTÉS PENDANT L'ANNEÉ 1929

(FIELD WORK CARRIED OUT DURING THE YEAR 1929)

By J. Morozewicz

Bulletin du Service Géologique de Pologne,
vol. 5, No. 3-4, 1930, p. LXXIII.

In Morozewicz's report on field work carried out in Poland during 1929 the following geophysical investigations made under the direction of the geophysicist E. W. Janczewski are mentioned:

1. In Kujawach, near Wloclawka. -- Seismic and gravimetrical methods were applied. Gravimetrical measurements were made at 137 points.

2. In the Carpathians (between Stryj, Bolechow, Rozniatow, Zawoj, Kalusz, and Turza Wielka).--Seismic method of prospecting was used.--W. Ayvazoglou.

(70) PROSPECTING WORK OF THE AZNEFT (IN RUSSIAN)

Editorial note

Azerbeidjanskoe Neftianoe Khoziaystvo,
vol. 10, No. 11, 1930, pp. 145-146.

The results of the geophysical prospecting work carried out during the years 1928, 1929, and 1930 are briefly summed up as follows:

1. Magnetic method: Magnetic anomalies in the region of Nefte-Chala proved to be in a distinct relation to the geological structure of the region. The dependance of these anomalies on the presence of magnetite in the Tertiary sediments is discussed.

2. Gravitational method: The maximum gravity in the region of Nefte-Chala is due to the presence of the lines limiting very dense and less dense rocks, as well as owing to the presence of faults.

Prospecting work on a large scale is to be carried out during 1931 mainly beyond the Baku region.

Electrical prospecting is to be tried out widely, owing to the fact that the results obtained by the gravimetrical and magnetic methods, applied during the past four years, have not been satisfactory.--W. Ayvazoglou.

(71) NOTE ON A COMPARISON OF SUNSPOT NUMBERS, TERRESTRIAL MAGNETIC ACTIVITY, AND LONG-WAVE RADIO SIGNAL STRENGTH

By L. W. Austin

Journal of the Washington Academy of Sciences,
vol. 20, No. 5, 1930, pp. 73-74.

A figure shows a comparison of the monthly averages of sunspot numbers, terrestrial magnetic activity (horizontal range) measured at Cheltenham, Md., and the daytime ratio field strength of signals received in Washington from transmitting stations at Bordeaux, France (FYL) (f = 19.5 kc., λ = 18,900 m.), and at Nauen, Germany (DFW) (f = 23.4 kc., λ = 12,800 m.).

The resemblance of the sunspot curve to the other three is not clear, but the similarity in the changes in magnetic activity to those in daylight signal strength seems to be unmistakable. The resemblance of the Bordeaux signal curve to that of the magnetic activity seems closer even than the resemblance between the two signal curves. The deep drop of both the magnetic and signal values in November (more rarely in December) is especially striking. This early winter drop in signals has often been noticed, and in the case of transmission between Europe and America, has been sometimes ascribed to the proximity of the signal paths to the area of Arctic darkness. It now appears that this and other seasonal variations both in magnetic activity and east-west long-wave signal strength may be due to common cases.--Author's note.

(72) ON ISOSTASY AND RELATED TOPICS

GEOLOGICAL SOCIETY IN WASHINGTON

Journal of the Washington Academy of Sciences,
vol. 20, No. 18, 1930, pp. 441-458.

Abstracts of the following five articles given before the Geological Society in Washington during the spring of 1930 are published:

1. Some problems of mountain structure and mountain history, by R. Longwell (pp. 441-446.)

2. Isostasy: What gravity measurements reveal, by G. R. Putnam (pp. 446-447.)

3. Some problems in isostasy, by R. W. Goranson (pp. 447-450).

4. Geotherms, by A. C. Lane, (pp. 450-454).

5. Isostasy from the geological point of view, by R. T. Chamberlin (pp. 454-458).--W. Ayvazoglou.

8. GEOLOGY

(73) PETROLEUM POTENTIALITIES OF GULF COAST PETROLEUM PROVINCE OF TEXAS AND LOUISIANA

By Donald C. Barton

Bulletin of the American Association of Petroleum Geologists, vol. 14, No. 11, 1930, pp. 1379-1400.

A new vista of the petroleum potentialities of the Gulf Coast petroleum province of Texas and Louisiana has been opened by the developments of the past few years. The coastal salt-dome area has been extended eastward to, and across, the Mississippi River. The area of good production has been extended southwestward to Refugio and may be expected to extend to the Rio Grande; it has been extended southward to Clay Creek, which lies in a hitherto practically nonproductive zone, and seems probably to have been extended eastward to Terrebonne Parish, La. The maximum depth of production has been extended to 7,444 feet, and the stratigraphic zone of good production has been extended down into the middle of the Claiborne. Deep and very deep salt domes have great potentialities, are being discovered in great numbers, but have certain drawbacks. The very deep salt ridges are unknown quantities. An enormously thick potentially productive stratigraphic section is present and offers great possibilities for production on such deep structures. A distinct tendency is shown for an increase of the Baumé gravity and gasoline content of the oil with increasing depth and in part with increasing stratigraphic depth. That change foreshadows a progressive change in the mean character of the oil of the future and gives the only suggestion, as yet, of a possible downward limit to production. The general magnitude of the recoverable reserves of oil in the Gulf Coast area of Texas and Louisiana at a shrewd guess is: surely at least, 3,500,000,000 barrels, probably at least 5,500,000,000 barrels and possibly at least 10,000,000,000 barrels.--Author's abstract.

(74) A STATISTICAL EXAMINATION OF THE SENSITIVITY OF A WATER TABLE TO RAINFALL AND IRRIGATION

By B. H. Wilsdon and R. Partha Sarathy

Memoirs of the Punjab Irrigation Research Laboratory, Lahore, vol. 1, No. 1, 1927, pp. 1-51.

The records collected in this article on a series of well observations in irrigated tracts of the Punjab from 1895 to 1926 constitute very valuable data on the sensitivity of a water table to rainfall and irrigation. In the summary of the first part of the article the authors say:

1. The analysis of well records and irrigation and rainfall statistics enables significant regressions to be established between the distribution of rainfall and irrigation throughout the year, and the annual fluctuations of the well levels.

2. The regression curve shows a constant characteristic in the occurrence of a maximum affect of irrigation and rainfall during the monsoon. A second maximum during the winter months is more variable in incidence as is also the occurrence of negative values of the regression.

3. For variations of the monthly irrigation and rainfall within the range of the standard deviations it is found that on the average approximately one-third of the water added to the surface of the soil during the year percolates to the water table.

4. The application of the regression curves in predicting the effect of restriction if irrigation is indicated.

5. A rough calculation shows that not more than one-third of the annual rise of the water table may be attributed to leakage from main canals and branches.

6. The slow changes in the well levels, when compared with the computed addition to the water table from irrigation rainfall and leakage from canals indicate that a well drained "doab" (land bounded by two rivers) is capable of maintained equilibrium. The steady rise in other canals is largely attributable to the absence of drainage, rather than to excessive percolation load.

In the second part of the article the authors give some calculations, the purpose of which is to obtain a mathematical curve showing the average effect of rainfall and irrigation in each month of the year on the well levels.

The necessary data for the calculation of the regression constants for each distributary are given in the following four tables:

Table I. Yearly values of the June or October series of:

(a) The observed depth of water.

(b) The polynomial value of this depth.

(c) The divergence (δd) of the observed from the polynomial value of D.

Table II. Give the constants of the polynomial fitted to W for the years shown.

Table III. Sume of squares and products.

Table IV. Matrix of co-factors.--W. Ayvazoglou.

(75) STATISTICAL STUDIES OF THE HISTORIES OF WATER-TABLES AS AFFECTED BY IRRIGATION AND RAINFALL

By B. H. Wilsdon and R. Partha Sarathy

Memoirs of the Punjab Irrigation Research Laboratory, Lahore, vol. 1, No. 2, 1928, pp. 1-24.

In the first part of the article the authors describe the results of an analysis of the relation between annual fluctuations of the water-table and the applied irrigation and rain water for a series of regions. The method adopted was:

1. To weight the data of rain-gage stations and observations wells, so that their records may be as truly representative as possible of the whole area.

2. To combine the weighted rainfall with the recorded irrigation in such a way that the total water applied, assumed uniformly distributed over the area W, may be represented as six independent variables of a fitted polynomial of the fifth order.

3. To eliminate secular changes from the weighted record of well fluctuations by fitting the best exponential curve.

4. To correlate the change in average well level, corrected for slow change, with the constants of the polynomial fitted to the W distribution.

The regression curves so obtained indicate a considerable variation throughout the year in the proportion of water which reaches the water-table from each application. Common features are a high proportion (one-third or more) during the monsoon when the humidity and intensity is high, and a more variable effect during the winter and early spring. The results in general confirm those obtained in a preliminary investigation by a similar method in a previous memoir but are statistically more reliable.

The numerical data are discussed and illustrated by diagrams.

In the second part of the article a detailed mathematical discussion of points arising in the analysis is given.

The necessary data for the calculation of the regression coefficient are given in the following four tables:

Table I gives the yearly values of δd (annual rate of rise of water-table) actual as well as those calculated from the regression formula with the exponential slow changes.

Table II gives the constants of the polynomial fitted to W for the year shown.

Table III. Sums of squares and products of the above-mentioned constants.

Table IV. Matrix of co-factors of the determinants in Table III.--W. Ayvazoglou.

(76) A STATISTICAL EXAMINATION OF THE DISCHARGE OF THE INDUS AT SUKKUR AND ITS RELATION WITH UPSTREAM SITES

By B. H. Wilsdon and R. Partha Sarathy

Memoirs of the Punjab Irrigation Research Laboratory, Lahore, vol. 1, No. 3, 1929, pp. 1-40.

The work described in this article was taken up at the request of the Indus Discharge Committee in September, 1928. The ultimate aim of the investigation has been defined as the determination of "the history of the water of the Indus and its tributaries during its passage through the Punjab and Sind to Sukkur." The desiderata of any acceptable formula were defined by this committee as follows:

(a) It must satisfy data available,

(b) more completely than any other formula, and

(c) must be capable of forecasting.

The sites at which discharge observations on the Indus and its tributaries were made are shown in a map.

In conclusion the authors emphasize the main deductions made from the statistical results, and distinguish lines along which further investigation appears profitable.

They say:

1. Regeneration appears to be quite definitely a real phenomenon determined by the time elapsed since high floods and to some extent by rainfall and associated humidity.

2. A formula has been developed which appears to be capable of predicting differences in discharge between upstream sites and Sukkur from a knowledge of the upstream discharges and the rainfall. This is of course not a prediction formula in the true sense. It affords hope of a true prediction formula being discoverable if reliable relations between rainfall in the catchments and discharges can be substantiated.

3. It is not at present possible to predict what effect a considerable disturbance in regime by a withdrawal will have at a downstream site although the absence of secular trend in recorded discharges proportionate to the very large increase in withdrawals which has taken place in the Punjab points to the ultimate establishment of regenerative conditions.

4. The probabilities of shortage in supply at Sukkur during the cold weather have been worked out on the assumption that the full Sind indent must be maintained. The withdrawals do not so greatly increase

the changes of shortage as to put the possibility of accommodation of the small Punjab demands by Sind completely out of court.

Full tabular matter has been collected in the appendix.--W. Ayvazoglou.

(77) A HYDRODYNAMICAL INVESTIGATION OF THE FLOW OF LIQUID IN A SATURATED POROUS MEDIUM SUCH AS A SOIL

By N. K. Bose and B. H. Wilsdon

Memoirs of the Punjab Irrigation Research Laboratory, Lahore, vol. 2, No. 1, 1929, pp. 1-17.

In the introduction to the article Wilsdon points out the difficulties, both theoretical and experimental, in studying the flow of liquids in porous media. The laws applied in hydrodynamics may be expected to aid in understanding and controlling the movements of a water-table, this subject being of great importance to the Punjab. Since the introduction of irrigation the water-table has risen over large tracts at an alarming rate. In some areas waterlogging on an extensive scale has resulted; in others, land once fertile has become saline and unculturable due to the nearness of the water-table to the surface. The further spread of both these evils, results of the same cause, calls for immediate and energetic remedial and preventive measures, neither of which can be confidently designed without considerable advances in theoretical knowledge. Moreover, without such knowledge it will be impossible to devise proper experimental methods. A brief review of applicable methods is therefore given in this article.

The second part of the article deals with applications of the mathematical treatment developed to ascertain simplified problems of practical importance. The mathematical work in this part is due to Bose. It has been constantly kept in view to obtain solutions in such a form that it will be possible to compare model experiments on viscous liquids with observations on the water-table.

Fundamental hydrodynamical equations applicable to the problems are deduced and the relation between the transmission constant of a soil and the viscosity of the model fluid is demonstrated.

Cases of percolation from a line course such as a canal, with and without distributed surface sources equivalent to the rainfall and irrigation load are discussed.

Unsteady motions corresponding with these cases are discussed.--W. Ayvazoglou.

9. NEW BOOKS

(78) Angenheister, G. Angewandte Geophysik (Applied Geophysics). Bearbeited von H. Haalck, W. Heine, J. N. Hummel, K. Jung, H. Martin, O. Meisser, H. Reich. XII and 556 pages with 253 figures. Price, stitched, R.M. 52; bound R.M. 54, 1930. Akademische Verlagsgesellschaft m.b.H., Leipzig. Contents: Geological bases of applied geophysics, by H. Reich; Gravimetric method of applied geophysics, by Karl Jung; Air-seismics and ground-seismics, by O. Meisser and H. Martin; Magnetic methods of applied geophysics, by H. Haalck; Electrical methods of applied geophysics, by J. N. Hummel and W. Heine; Practical application of electrical methods, by W. Heine; Radioactive methods, by J. N. Hummel.

(79) Haarmann, Erich, Prof. Dr. Die Oszillationstheorie. Eine Erklärung der Krustbewegung von Erde und Mond (The Oscillation Theory; An Explanation of the Movement of the Crust of the Earth and of the Moon). 260 pages, 78 figures, 1 table. Verlag R. Enke, Stuttgart 1930. Price, stitched, R.M. 17.00, bound, R.M. 19.00.

(80) Hermanns, Hubert. Techno-Diktionar, Deutsch-Englisch-Italienisch. (Technical dictionary, German-English-Italian). 432 pages. Berlin, Hubert Hermanns, 1929. Price, $3.75.

(81) Joly, John. The surface history of the earth. 2nd edition. 211 pages, figures 11, map. Oxford press, 1930. Price, $3.50.

(82) Kraus, E. H., and Hunt, W. F. Tables for the determination of minerals by means of their physical properties, occurrences, and associates. 2nd edition. IX and 266 pages. McGraw-Hill Book Co., New York, 1930. Price, $3.00. The tables of this edition contain the names of a few more minerals and varieties.

INDEX[1]

1 - The first figure refers to the number of the abstract; the second to the method of prospecting as indicated in the Table of Contents, and the third to the page.

I. C. 6453

June, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

THALLIUM

BY

ALICE V. PETAR

I. C. 6453.
June, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

THALLIUM[1]

By Alice V. Petar[2]

CONTENTS

INTRODUCTION

Thallium is one of the rare metals that has come into commercial use within recent years. Its comparative scarcity, combined with a resemblance to the common metal lead, have retarded the development of uses for thallium. Although production is still measured in pounds rather than in tons, it is now finding application in a number of fields. Perhaps the earliest commercial use of thallium was as a constituent of optical glass of higher refractive power than similar glasses containing lead. One of the later applications to attract attention was the use of the oxysulphide in a sensitive electrical device similar to the selenium cell. The "Thalofide" cell, which was patented in 1919, constituted for a time the most important outlet for thallium. During the next few years the limited use of thallium and its salts in other fields was insufficient to absorb an annual production of a few hundred pounds, and the price dropped from $16 or $18 per pound in 1922 to $5 per pound in 1924. Beginning in 1925 several new uses were developed, especially as a poison for rodents and ants, and as a result the demand for thallium increased sufficiently to cause a recovery in price to the present level of from $12 to $15 per pound.

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1 - The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6453."

2 - Rare metals and nonmetals division, U. S. Bureau of Mines.

There are no commercial ores of thallium, although it is present in small quantities in a great many minerals. The supply was formerly obtained solely from flue dusts that accumulate in sulphuric acid plants where pyrite is used. The thallium content of these flue dusts is small, -- usually about half of 1 per cent, -- and the thallium is produced only as a by-product. According to information contained in a letter from L. G. Matthews, of the American Smelting and Refining Co., to the Bureau of Mines, at present most of the domestic supply is a by-product of the purification of cadmium, itself a by-product of the smelting of other metals.

DESCRIPTION AND PROPERTIES

The physical and chemical properties of thallium are very similar to those of lead. The metal is white with a bluish-gray tinge, somewhat paler than lead. It has a bright metallic luster when freshly cut but dulls quickly when exposed to air. Thallium is softer than lead; it can be scratched by the finger nail and is easily cut with a knife. It is malleable but has little tenacity, and can be squeezed (extruded) but not drawn, into wire. A number of determinations have been made for the melting point of thallium, ranging from 204.05 to 303.7°C., but 303.5°C. has been accepted by the International Critical Tables. According to A. Lamy the specific gravity of thallium is 11.862, although slight variations from this figure have been given by other investigators. The metal has a crystalline structure and when bent gives forth a sound like the "cry of tin." Thallium amalgamates with mercury and forms alloys with many other metals. It is slowly soluble in alcohol, insoluble in liquid ammonia, and combines directly with sulphur, phosphorus, and the halogens.

HISTORY

The discovery of thallium dates back to the early part of 1861, when W. Crookes noted a bright line in the spectrum of the selenium residues from a sulphuric acid factory in the Harz Mountains. He was looking for tellurium, but spectroscopic examination revealed the presence of an unknown element, to which he gave the name "thallium," - "from the Latin thallus, a budding twig - a word which is frequently employed to express the beautiful green tint of young vegetation; and which I have chosen because the green line which it communicates to the spectrum recalls with peculiar vividness the fresh color of vegetation in spring." He announced the discovery on March 30, 1861, in a paper "On the Existence of a New Element, Probably of the Sulphur Group." Crookes was at first hampered by a lack of material, but in the course of a year he was able to collect a few grains of the metal in powder form.

In May 1862 A. Lamy observed the green spectral line from the chamber deposit of a sulphuric acid plant where Belgian pyrites were roasted. Lamy obtained several hundred grams of the metal and was able to make considerable progress in determining the chemical and physical properties of the new element. Subsequently H. L. Wells, S. L. Penfield, and other investigators studied the metal and its properties. Thallium continued as an object of scientific interest, but only within the last decade have its commercial possibilities been recognized.

OCCURRENCE

Four very rare minerals contain appreciable percentages of thallium. Crookesite, a thallium-copper-silver-selenide, $(CuTlAg)_2Se$, which contains from 16 to 19 per cent of thallium, is found in Skrikerum, Sweden. Lorandite, a sulphide of thallium and arsenic, $TlAsS_2$, contains from 59 to 60 per cent of thallium and is found in Macedonia. Hutchinsonite, a sulphoarsenide of thallium, lead, silver, and copper approximating $(Tl,Ag,Cu)_2S.PbS.2As_2S_3$, has a thallium content of from 18 to 25 per cent. This mineral occurs in white dolomite of the Lengenbach Quarry in the Binnental, Switzerland. Vrbaite, found in Macedonia, is a sulphide of thallium, arsenic, and antimony $(TlAs_2Sb_5)$ which contains 29 to 32 per cent of thallium.

Thallium is also present in small quantities in a great variety of rocks and minerals, such as pyrites, zinc blende, hematite, lepidolite, muscovite, orthoclase, pitchblende, berzelinite, frenzelite, pyrolusite, manganite, carnallite, etc. Owing to its occurrence in some of the common minerals, thallium is found in commercial products such as zinc, cadmium, platinum, bismuth, tellurium, and sulphuric acid. Copper pyrites and iron pyrites frequently contain thallium, and the flue dusts from sulphuric acid plants where thalliferous pyrites are burned have constituted one of the principal sources of supply. The thallium content of such flue dusts is usually less than half of 1 per cent, but in some instances a content as high as 7 per cent has been noted.

USES

Alloys of lead and thallium are somewhat unusual in that they have higher melting points than either of the component metals; they are used in rather small quantities in special types of electrical fuses. An alloy containing 10 per cent thallium, 20 per cent tin, and 70 per cent lead is resistant to the corrosive action of mixtures of sulphuric, nitric, and hydrochloric acids. This alloy has been recommended for use as an anode for the electrolytic deposition of copper, since its corrosion is less than one-fifth that of lead alone.[3]

In a discussion of the possibility of using lead-base bearing metal in place of tin-base, O. W. Ellis[4] reports that the results of experiments indicate that the addition of thallium to lead-base alloys markedly improves their resistance to deformation; and that yield point and ultimate strength tests show that an alloy containing lead 72, antimony 15, tin 5, and thallium 8, is superior to the best tin-base bearing alloy.

Silver alloys containing as much as 22 per cent thallium have been patented by I. G. Farbenind A.-G.[5]

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3 - Hopkins, B. Smith, Chemistry of the Rarer Elements: D. C. Heath & Co., New York, N. Y., 1923, pp. 125-126.
4 - Ellis, O. W. Increasing the strength of lead-base bearing metals. Am. Metal Market, vol. 35, 1928, pp. 1-4, 44.
5 - I. G. Farbenind, A.-G., Silver alloys resistant to chemical action. Brit. Pat. 297,665, June 30, 1927.

Many uses have been found for thallium compounds. By far the most important commercial outlet for the element is the utilization of thallium sulphate as a poison for rodents and as an insecticide for ants. The sulphate has the advantage of being tasteless and odorless. Ernest G. Enck[6] has described several formulas that are said to be effective as poison bait for field mice. In one, $1\frac{1}{2}$ pounds of thallium sulphate are dissolved in 6 quarts of boiling water, and $\frac{1}{2}$ pound of starch, which has been mixed in a little cold water, is added. The mixture is boiled two or three minutes, then $\frac{1}{2}$ pint of glycerine is added, and the boiling is continued for a few minutes longer. In another method $2\frac{1}{2}$ per cent of thallium sulphate is added to a tapioca flour and made into a paste, which is spread on sliced bread. A third recipe calls for $\frac{1}{4}$ ounce of thallium sulphate, which is dissolved in a large teacup of water and brought to a boil. To this is added $\frac{1}{2}$ cup of corn syrup and 12 ounces of peanut butter, which may be spread on two ordinary sized loaves of bread sliced quite thin. The slices are then cut in small squares. It is said that noticeable results should be seen at about the third day after placing.

A syrup prepared by boiling together 1 pint of water, 1 pound of sugar, 27 grains of thallium sulphate, and 3 ounces of honey, is said to be effective in exterminating a species of red ants which arsenic syrups failed to control. The thallium appears to act as a slow cumulative poison. In a number of houses and apartments the entire colony was destroyed within three weeks to a month. The pavement ant is even more readily controlled.[7]

Because of their high refracting power, thallium compounds are used in the manufacture of certain kinds of optical glass in which a high refractive index is required.

The oxysulphide of thallium is used in the "Thalofide" cell, which is more sensitive to light, especially that of low intensity and long wave lengths, than the selenium cell. Its electric resistance drops 50 per cent on exposure to a quarter-foot candle. Several patents for the "Thalofide" cell were issued to Theodore W. Case in 1919 and 1920. Mr. Case has also patented a compound of thallium and bromine (U. S. Patent 1,342,842, June 8, 1920) which is used as an electric resistance material in the Bellphotophone or similar apparatus.

The use of thallium and its compounds to prevent knocking in internal combustion engines has been patented.[8] According to the patent specifications, thallium or thallium oxide may be vaporized outside the cylinder by an electric arc or a thallium compound may be mixed with the fuel or otherwise injected into the engine. The compounds which may be used include thallium ethyl, benzylate, phenylethylate, oleate, amylalcoholate, and acetoacetate.

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6 - Enck, Ernest G., Thallium and its Uses: Foote-Prints, vol. 2, No. 1, 1929, pp. 15-17.

7 - Popenoe, C. H., Thallium as an Insecticide: Science, vol. 64, 1926, p. 525.

8 - Asiatic Petroleum Co., Ltd., and Egerton, A. C., Use of thallium and its compounds to prevent knocking in internal combustion engines. Brit. Pat. 279,560, July 29, 1926.

A liquid amalgam containing 8.5 per cent of thallium has been used in thermometers for recording temperatures as low as -60°.

Thallous chloride has found use as a "getter" in tungsten lamp to prolong the life of the filament. It is one of the few lower chlorides is more stable than the compound of the higher state of oxidation.

In a study of heavy liquids for mineralogical analyses[9] the Bureau of Mines found that the most suitable liquid for sink-and-float work on minerals of high specific gravity was a water solution of the double thallous formate-malonate. This double salt, with a specific gravity of 4.9, melts at 60° C. and is miscible in all proportions with water. At room temperatures the salt is soluble enough to give a solution with a specific gravity of nearly 4.3. As a liquid of intermediate gravity, an aqueous solution of thallous formate is recommended. It is less expensive than the formate-malonate. A maximum specific gravity of 3.5 is reached at room temperatures, and gravities up to 4.95, the gravity of the molten salt, may be obtained.

Thallium acetate has been used for the treatment of certain kinds of tuberculosis and ringworm. At one time it was used as a depilatory.

PRODUCTION

At present thallium is produced in the United States by only one company, the American Smelting & Refining Co., which recovers the metal as a by-product in its cadmium operations.

MARKETS AND PRICES

The demand for thallium and its salts is not large. The only compound of thallium that has attained any marked commercial development is the sulphate, and it is understood that this demand is being taken care of largely by importations from Germany. No figures covering imports are available.

Early in 1931 thallium metal was quoted at from $12.50 to $15 per pound, and the sulphate was selling at prices ranging from $10 to $15 per pound.

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9 - Sullivan, John D., Heavy Liquids for Mineralogical Analyses: Tech. Paper 381, Bureau of Mines, 1927, 25 pp.

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Scott Turner, Director

INFORMATION CIRCULAR

REVISION OF THE FREE ENERGY OF FORMATION OF SULPHUR DIOXIDE

BY

E. D. Eastman

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INFORMATION CIRCULAR

A DEPARTMENT OF COMMERCE - BUREAU OF MINES

A REVISION OF THE FREE ENERGY OF FORMATION OF SULPHUR DIOXIDE[1]

By E. D. Eastman[2]

The recent determination by Eckman and Rossini[3] of the heat of formation of sulphur dioxide appears to be of much greater accuracy than any previous measurement. The value obtained by them, 70,940 cal., is in good agreement with that of Thomsen (71,080), but differs by about 1,700 calories from Berthelot's result (69,260), and by 1,940 calories from the provisional figure (69,000) adopted by Lewis and Randall[4] in their studies of the free energies of the sulphur compounds. This large discrepancy suggested the desirability of a recalculation of the free energy of sulphur dioxide. In addition it is now possiblt to fix the specific heat of S_2 gas with greater certainty than at the time of the original calculations. These two changes are incorporated in the recalculations described below, and directly affect the values of ΔH_0 and the $\Delta\Gamma$ terms in the free energy equation. In the determination of the integration constant of the latter equation, the free energies and specific heats of water vapor and the oxides of carbon are required as auxiliary data. In this work these are based, with certain exceptions, upon the equations previously deduced,[5,6] which differ appreciably from the corresponding equations of Lewis and Randall.

The principal equilibria employed below, and the plan of calculation, remain the same as in the calculations of Lewis and Randall. Before beginning this revision a search of the literature was made, extending to August, 1930, for studies of additional equilibria. Only one reaction was found that gives promise of increased accuracy. This is the equilibrium of sulphur dioxide with copper, cuprous oxide, and cuprous sulphide. In this case, however, certain of the subsidiary data are lacking, and until they become available, the existing studies of the equilibrium in this system can not be successfully utilized. The changes in the numerical data mentioned above, especially in the thermochemical constants, are therefore solely responsible for the rather significant differences of the revised from the original values of the free energy.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6454."

2 Consulting physical chemist, Metallurgical Division, U. S. Bureau of Mines.

3 Eckman, J. R., and Rossini, Frederick D., The Heat of Formation of Sulphur Dioxide: Bureau of Standards Jour. of Research, vol. 3, 1929, pp. 597-618.

4 Lewis, G. N., and Randall, M., Thermodynamics and the Free Energy of Chemical Substances: McGraw Hill Book Co., New York, 1923, 1st ed., 653 pp.

5 Eastman, E. D., The Free Energy of Water, Carbon Monoxide, and Carbon Dioxide: Inf. Cir. 6125, Bureau of Mines, May, 1929, 15 pp.

6 Eastman, E. D., Specific Heats of Gases at High Temperatures: Tech. Paper 445, Bureau of Mines, 1929 27 pp.

FORMULATION OF THE FREE ENERGY EQUATION

Specific Heat Data

The specific heats required in these calculations have been expressed by the following equations:

$$S \text{ (rhomb.)};\quad C_p = 4.12 + 4.7 \times 10^{-3}\,T$$
$$S_2(g);\quad C_p = 8.30 + 0.3 \times 10^{-3}\,T$$
$$O_2;\quad C_p = 6.76 + 0.606 \times 10^{-3}\,T + 0.13 \times 10^{-6}\,T^2$$
$$SO_2(g);\quad C_p = 7.70 + 5.30 \times 10^{-3}\,T - 0.83 \times 10^{-6}\,T^2$$

The equation for rhombic sulphur is that given by Lewis and Randall. For oxygen and sulphur dioxide the equations are those previously derived.[7] It will be recalled that for lack of detailed data sulphur dioxide was assumed to be identical with carbon dioxide in its specific heat. This is probably not as nearly correct as might be desired, and when more accurate equilibrium data are obtained this equation should be reconsidered. The only case requiring extended comment here, however, is that of S_2 gas, which will now be discussed.

Lewis and Randall took the heat capacity of S_2 the same as that of O_2. In the previous paper on specific heats,[8] however, it was pointed out that many of the more loosely bound diatomic molecules take up appreciable quantities of vibrational energy at relatively low temperatures. It was pointed out there that S_2 was more nearly like Cl_2 in this regard than any of the other gases, and might be roughly classed with it. A more exact consideration of the course of its specific heat curbe shows appreciable differences from the chlorine equation and seems to require separate treatment.

When the characteristic frequency of vibration of a diatomic molecule in the lowest state is known, it is possible to represent its vibrational specific heat approximately by means of the corresponding "Einstein function." In the case of S_2 the first quantum jump in vibration is approximately known, at least, from the calculation of Birge[9] based on spectral data. Taking this jump as 0.089 volt-equivalent, the corresponding Einstein function gives the values in Table 1 for vibrational heat capacity. The total heat capacities given there are then obtained by adding these values to 7/2 R, the molal heat capacity at constant pressure of a diatomic gas with rotational degrees of freedom fully excited.

Table 1.- Calculated heat capacity at constant pressure in calories per mol, of S_2 gas

T	C vibrational	C_p total	C_p equation	C_p L. & R.
250	0.56	7.52	8.38	6.75
300	0.80	7.76	8.39	6.80
500	1.41	8.37	8.45	7.00
750	1.70	8.66	8.53	7.25
1,000	1.82	8.78	8.60	7.50
1,500	1.90	8.86	8.75	8.00
2,000	1.94	8.90	8.90	8.50
2,500	1.96	8.92	9.05	9.00

7 Eastman, E. D., Specific Heats of Gases at High Temperatures: Tech. Paper 445, Bureau of Mines, 1929, 27 pp.

8 See footnote 7.

9 Birge, Raymond T., International Critical Tables: vol. 5, McGraw-Hill Co., New York, 1929,

As will be seen from the table, the heat capacity of S_2 is rapidly varying in the temperature range immediately above 300°K. The form of this variation, moreover, is not representable over wide temperature intervals by simple equations of the usual algebraic type. One must therefore choose, if this type of equation is retained, between some sacrifices of accuracy or an extremely cumbersome equation. In this case the first course has been followed, since only moderate accuracy may be claimed in other portions of the work. The equation given above results in values of heat capacity shown in Table 1 under the heading C_p equation. At 300°K. the value from the equation differs by 8.1 per cent from the value assumed to be correct. At 500° this difference is reduced to 1 per cent; and in the interval 600 to 2,000° it never exceeds 2 per cent. This equation also should be revised when more accurate equlibrium data are available, but is believed adequate for the present purpose. For comparison, values calculated from the equation used by Lewis and Randall are shown in the last column of the table.

Heats of Reaction

The reactions in which we are interested are

$$1/2\ S_2(g) + O_2 = SO_2\ (g) \qquad \text{I}$$

and

$$S\ (\text{rhomb.}) + O_2 = SO_2\ (g) \qquad \text{II}$$

The value of ΔH in reaction II, (ΔH(II)), was directly determined at 298°K. by Eckman and Rossini as -70,940 cal. To obtain ΔH(I) at 298°, Lewis and Randall resorted to an indirect determination through a value calculated from free energy changes in the reaction

$$2\ S(\text{rhomb.}) = S_2\ (g) \qquad \text{III}$$

Using the specific heat equations given above, we have

$$\Delta C_p(\text{III}) = 0.06 - 9.1 \times 10^{-3}T$$

$$\Delta H(\text{III}) = \Delta H_0(\text{III}) + 0.06\ T - 4.55 \times 10^{-3}T^2$$

and

$$\Delta F°(\text{III}) = \Delta H_0(\text{III}) - 0.06\ T \ln T + 4.55 \times 10^{-3}T^2 + I_1T \qquad (1)$$

From the values of $\Delta F°$(III) obtained by Lewis and Randall at 718° and 298°K., namely 3,320 cal. and 18,280 cal., both ΔH_0(III) and I_1 may be calculated. The resulting values are ΔH_0 = 29,840 cal. and I_1 = -39.81. Lewis and Randall, taking the heat capacity of S_2 the same as of O_2, found ΔH_0 = 30,580 cal., and I_1 = -52.4 ΔH_{298}(III) is calculated from the above values to be 29,454 cal., as compared with 29,690 cal. found by Lewis and Randall.

Combining ΔH(II) and ΔH(III) as found above gives ΔH_{298}(I) = -85,667 cal. From the equation

$$\Delta C_p(\text{I}) = -3.21 + 4.544 \times 10^{-3}T - 0.96 \times 10^{-6}T^2,$$

it is seen that

$$\Delta H(\text{I}) = \Delta H_0(\text{I}) - 3.21\ T + 2.272 \times 10^{-3}T^2 - 0.32 \times 10^{-6}T^3.$$

Employing ΔH_{298}(I) found above, ΔH_0(I) becomes -84,905 cal.

The Free Energy Equation

The various data obtained above may be combined to give the free energy equation for reaction I, i.e., for the formation of SO_2 gas from S_2 gas and oxygen. It is

$$\Delta F°(I) = -84905 + 3.21\ T \ln T - 2.272 \times 10^{-3}T^2 + 0.16 \times 10^{-6}T^3 + I_2T \qquad (2)$$

The integration constant, I_2, of this equation is then to be determined from the equilibria discussed below.

DETERMINATION OF THE INTEGRATION CONSTANT

The first equilibrium employed by Lewis and Randall for the determination of I_2 is that between water vapor, liquid sulphur, hydrogen sulphide, and sulphur dioxide at the boiling point of sulphur. From their studies of this equilibrium they obtained the free energy change at this temperature in the reaction

$$2H_2O(g) + 3/2\ S_2(g) = 2H_2S(g) + SO_2(g). \qquad IV$$

Their value is $\Delta F°_{718}(IV) = 2{,}250$ cal. Calculating from their equation XXXVIII, - 22 the free energy of $H_2S(g)$ at 718°, one finds -12,860 cal. The free energy of water vapor, from the equation previously referred to[10], is -49,293 cal. at 718°. These values combined give the free energy of SO_2 at 718° as -70,614 cal. The latter value substituted in equation (2), leads to the value $I_2 = 0.34$.

The second calculation of I_2 is based upon the equilibrium studies of Randall and Bichowsky. From their work Lewis and Randall have derived values of the equilibrium constants, K(V) in the reaction

$$3H_2 + SO_2 = H_2S + 2H_2O. \qquad V$$

Combining the equations already referred to of Lewis and Randall for the free energy of H_2S, and that of Eastman for H_2O, leads to an expression for I_2 as follows:

$$I_2 = R \ln K(V) - 48750/T + 1.75 \ln T + 4.355 \times 10^{-3}T - 0.882 \times 10^{-6}T^2 - 2.83$$

With the values of log K(V) from Table 3 of Lewis and Randall[11] the following values of I_2 are found:

T	1160	1362	1473	1473	1645
I_2	-1.49	-1.87	-2.29	-2.62	-3.15

The third calculation utilizes the data obtained by Ferguson in his study of the reduction of SO_2 by CO. Lewis and Randall write the reaction

$$2CO_2 + 1/2\ S_2 = 2CO + SO_2 \qquad VI$$

10 Eastman, E. D., The Free Energy of Water, Carbon Monoxide, and Carbon Dioxide: Inf. Cir. 6125, Bureau of Mines, May, 1929, 15 pp.

11 Lewis, G. N., and Randall, M., Thermodynamics and the Free Energy of Chemical Substances: McGraw Hill Book Co., New York, 1923, p. 546.

and summarize Ferguson's data by means of two values of log K(VI), namely, -3.56 at 1,275°K. and -2.38 at 1,460°K. The equation for the calculation of I_2, obtained by combination of equation 2 with the equations for the free energies of CO and CO_2, is

$$I_2 = -R \ln K(VI) - 49150/T + 6.93 \ln T - 13.118 \times 10^{-3}T + 4.204 \times 10^{-6}T^2 - 6.516 \times 10^{-10}T^3 - 15.68$$

From the values above of log K(VI), I_2 is found to be 0.37 and -0.22 at the two temperatures given.

The data respecting I_2 from the three sources above is summarized in Table 2.

Table 2.- Comparison of values of the integration constant, I_2, of the free energy equation

Reaction	Average I_2	Deviation	Trend
Sulphur and water at 718°K.	0.34	0.6	-
Sulphur and water, 1,160 to 1,645°K.	-2.28	.3	1.7
Reduction of SO_2 by CO, 1,275 and 1,460°K.	0.15	0.6	0.6

The "deviations" in Table 2 are estimates of the approximate deviation of results at a single temperature from their mean, while the "trend" shows the extreme variation of values at different temperatures.

The agreement between the first and third values of I_2 in Table 2 is about the same as in the calculations of Lewis and Randall, while the second is a little further removed from the other two. The trends in I_2 in the last two series are also slightly increased. The remark of Lewis and Randall that, considering the difficult nature of the equilibrium measurements the agreement is surprisingly good, still applies. This is especially true in view of some obvious uncertainties that still remain in the subsidiary data. One of the more important of these is in the heat of formation and specific heat of H_2S, the free energy of which enters in several places in these calculations. The heat of formation was calculated by Lewis and Randall from equilibrium measurements at high temperatures. The specific heats of S_2 gas and H_2S used by them in their calculations are both too low. These errors are at least partly compensating, and judging from such tests of the results as may be applied, may be very closely compensated. This has been assumed to be the case in the present calculation, no attempt being made to improve the ΔC_p values for this substance. For more accurate work this should, however, be considered, as should also those for SO_2.

Of the three sets of measurements above, the first represents the smaller extrapolation from the standard temperature, while the third is perhaps more direct and less affected by multiple errors than the others; both seem preferable to the second series. The value of I_2 in equation (2) is therefore chosen as 0.25, giving some weight to both the first and third methods, but excluding the second.

This value of I_2 used in equation (2) gives $\Delta F° = -79,580$ at 298°K. as the free energy of formation of SO_2 from S_2 gas. Combined with equation (1), its free energy from rhombic sulphur is found to be -70,440 cal.

SUMMARY OF RESULTS

The recalculations described above lead to the following revised equations:

$2\ S(rhomb.) = S_2(g);$ $\Delta F°_{298} = 18{,}280$

$$\Delta F° = 29{,}840 - 0.06\ T \ln T + 4.55 \times 10^{-3}T^2 - 39.81\ T$$

$1/2\ S_2(g) + O_2 = SO_2(g);$ $\Delta F°_{298} = -79{,}580$ cal.

$$\Delta F° = -84{,}905 + 3.21\ T \ln T - 2.272 \times 10^{-3}T^2 + 0.16 \times 10^{-6}T^3 + 0.25\ T$$

$S(rhomb.) + O_2 = SO_2(g);$ $\Delta F°_{298} = -70{,}440$ al.

The values recorded by Lewis and Randall for the above reactions are, respectively, 18,280, -78,560 and -69,660. (There is, however, a discrepancy in these values. If the first two are correct, the third apparently should be -69,420.)

The above values for sulphur dioxide are recommended for temporary and immediate use, but may be subject to further correction when the thermodynamics of the reaction $Cu_2S + 2Cu_2O = 4Cu + SO_2$ has been satisfactorily discussed. Work on certain phases of this reaction is now in progress at the Pacific Experiment Station of the U. S. Bureau of Mines.

I. C. 6455 June, 1931

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

ZIRCONIUM

PART I. GENERAL INFORMATION

BY

E. P. YOUNGMAN

I. C. 6455.
June, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

ZIRCONIUM[1]

PART I. GENERAL INFORMATION

By E. P. Youngman[2]

TABLE OF CONTENTS

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2 - Rare metals and nonmetals division.

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INTRODUCTION

Before the World War the use of zirconium and its compounds was largely experimental and confined almost wholly to Germany and Austria. Occasionally small lots of zirconium ore had been produced in the United States (1,000 pounds in 1869, 26 tons in 1883, and 3,000 pounds in 1903) and used (with monazite) in the manufacture of various lighting devices, especially incandescent gas mantles; but it was not until 1906 that the discovery of the natural zirconium oxide in large quantities near Sao Paulo, Brazil, first gave promise that a zirconium industry eventually might be developed. Edward Rietz, who worked the Brazilian deposits, interested German chemists in extending the use of zirconium, and by 1911 trade journals were listing German firms as manufacturers of compounds or products derived therefrom. For a time zirconia was one of the various oxides used in the Nernst lamp. It was used also for crucibles and hearths as refractory material; in the manufacture of chemical utensils; in enamels; in the place of bismuth subnitrate for defining Röntgen-ray pictures of the stomach; as a pigment; in medicine; and in a number of other ways. The Germans were credited also with producing a remarkable zirconium steel, which, it was claimed, was much superior to other alloy steels.

During the World War the United States Government investigated the possibilities of zirconium, as well as those of other steel-hardening elements, and a large automobile manufacturer actually purchased a substantial quantity of ore with the purpose of employing zirconium in automobile steels. Immediately after the armistice the Government ceased its investigations; but despite the difficulties experienced by various investigators in reaching concordant results, interest in the use of zirconium as a steel-hardening agent, as well as in other uses of the element and its compounds, continued. In 1918 the Foote Mineral Co., which had been interested in zirconium for several years, was exploiting the deposit in Brazil formerly worked by the Germans and was supplying zirconia in the form of ore, brick, and cement for refractory and other purposes. About the same time the Electro Metallurgical Co., which before the entrance of the United States into the war was experimenting with zirconium alloys, supplied in tonnage quantities the demand for these alloys. In 1919 it was reported that American manufacturers, after several years of research, were producing in commercial quantities pure zirconium oxide, which was finding a place in the manufacture of small refractory articles, scorifying dishes, crucibles, tubes, and one-piece furnace linings.

In 1920 the lowering of the price of zirconium metal powder (because of improved metallurgical methods) promised the production of the metal on a semicommercial scale. In 1924 the Titanium Alloy Manufacturing Co. introduced a new opacifying agent, "Opax," made from zircon. A year or two later some of the large pottery manufacturers became interested in the use of zirconium oxide as an opacifying agent, the increased production of titanium pigments having resulted in the production of zirconium as a by-product at a price that could compete with tin oxide. In 1928 it was reported that the industrial use of zirconium oxide was increasing steadily in the manufacture of brick, crucibles, and other refractory products, such as high-temperature cements. However, the reluctance of producers to install additional and more suitable

machinery for sorting and concentrating the ore was limiting somewhat the manufacture of these products. In 1929 the report was that, although a shortage of high-grade ore existed, the use of zirconium in enamels for opacifying and its use as a refractory material had increased. In 1930 the Foote Mineral Co. reported greatly increased use of zirconium oxide as a refractory.

DESCRIPTION AND PROPERTIES

Zirconium falls into Group IV of the periodic system. Its position in this group, previously indicated by its analogies with the elements of the fourth group, was first confirmed by means of the high-frequency spectrum. As might be expected from their relative positions in the periodic table, zirconium resembles titanium in many of its characteristics and has properties in common with germanium also.

As zirconium is an exceedingly difficult metal to isolate, conflicting statements have been made with respect to its nature and properties; and, according to Lee,[3] most of the data with respect to zirconium, painfully acquired during a century and a half, must now be revised as a result of the discovery that the substance formerly considered an element is really a mixture of two elements, one zirconium and the other hafnium.

Description of the Metal

In a reasonably pure condition (as far as it has been prepared), 99.69 to 99.89 per cent, the metal is known as an amorphous black powder, which is steel-gray in color when melted, and which gives a lustrous metallic mirror when burnished. The existence of crystalline and graphitic forms has been claimed but not proved, according to Venable,[4] who states that investigators reporting them had alloys rather than the pure metal at hand. Marden and Rich seem to agree with this conclusion of Venable. In discussing the different forms of the metal, they refer to the "so-called" crystalline zirconium prepared by Troost and refer also to the method for the preparation of the graphitic zirconium (first used by Troost) as a doubtful one for the production of a pure metal.[5] According to these same authors,[6] the coherent (or sintered) metal shows decided crystalline structure under the microscope, but many of the references in the literature to crystalline zirconium describe the aluminum alloy.

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3 - Lee, O. Ivan, The Mineralogy of Hafnium: Chem. Reviews, vol. 5, No. 19; Am. Chem. Soc., 1928, p. 35.

4 - Venable, Francis P., Zirconium and its Compounds: Am. Chem. Soc. Monograph Ser., The Chemical Catalog Co. (Inc.), New York, N. Y., 1922, p. 26.

5 - Marden, J. W., and Rich, M. N., Investigations of Zirconium with Especial Reference to the Metal and the Oxide: Bull. 186, Bureau of Mines, 1921, pp. 36-37.

6 - Marden, J. W., and Rich, M. N., Work cited, p. 37.

Properties of the Metal

The melting point of zirconium is 2,130°C. The specific gravity of hafnium-free zirconium is 6.52, and its atomic volume is 13.97.[7] The atomic weight of the metal is 90.6; its hardness, 6.7; its specific refraction, 0.242; and its atomic refraction, 21.9.[8]

Pure zirconium rods produced by thermal decomposition of zirconium tetraiodide can be worked, although rods made from zirconium powder can not be. The workable rods may be hammered, rolled out, and drawn to thin filaments, having a diameter of less than .0008 inch.

Chemically, zirconium is a very reactive metal, It is readily attacked by hydrofluoric acid, hot concentrated sulphuric acid, and aqua regia. Concentrated hydrochloric acid, however, acts upon it only slightly and nitric acid not at all. It combines with all gases, except the rare gases. Amorphous zirconium when heated below 700° absorbs hydrogen, forming a solid hydride, which is fully dissociated at 800°. Zirconium combines readily with oxygen, burning with a bright light when heated even much below red heat; the denser and purer metal oxidizes only slowly when heated in air up to 270°. Amorphous zirconium readily combines with nitrogen; and when heated with carbon it forms a carbide, which if rich in carbon is gradually decomposed in air but which otherwise remains stable. At high temperatures, the metal readily combines with silicon and boron. Zirconium when heated and acted upon by chlorine and bromine forms tetrahalides; when heated and passed over by ammonia it forms the nitride. Sulphur combines with heated zirconium. Red phosphorous combining with zirconium gives a black powder. Zirconium forms alloys with copper, silver, aluminum, metals of the iron group, and others, but not with tin or lead.[9]

In a comparatively recent contribution to the trade literature, De Boer[10] says of zirconium metal, especially of that produced by his own processes, that it has a specific electrical resistance, at 0°C., of 41 times 10^{-6} ohms per cm^3; a temperature coefficient of resistance, at 0 to 100°C., of 0.0044; a coefficient of linear thermal expansion, at room temperature, of 6.3 times 10^{-6}; and an electron emission of 1.3 milliamperes per square centimeter at 1,700° K. Zirconium has about the same electron emission at 1,700° K. as thorium at 1,400° K. and tungsten at 2,000° K. (See also article by De Boer referred to in footnote 7.)

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7 - De Boer, J. H., Zirconium: Foote-Prints on the Rare Metals and Unusual Ores, vol. 3, No. 2, Foote Mineral Co. (Inc.), Philadelphia, Pa., 1930, p. 18.

8 - Venable, Francis P., Work cited, pp. 27-28.

9 - Venable, Francis P., Work cited, p. 30.

10- De Boer, J. H., Zirconium: Ind. Eng. Chem., vol. 19, No. 11, Am. Chem. Soc., Nov., 1927, pp. 1256-1259.

Description of the Oxide

The only definitely known oxide of zirconium is the dioxide, which occurs in nature as baddeleyite, and which can be made in the laboratory by the ignition of different zirconium compounds, as the sulphate, the oxychloride, and the hydroxide. According to Marden and Rich,[11] zirconia (ZrO_2) exists in two forms, amorphous and crystalline. The amorphous zirconia is a white powder of flourlike appearance when finely pulverized and is exceedingly inert toward chemical agents. This is the form that is generally produced in the laboratory or in industrial plants. Crystalline zirconia, according to De Boer,[12] has at least three crystal modifications (monoclinic, pseudoregular-tetragonal, and pseudohexagonal-rhombic forms).

Properties of the Oxide

As is the case with the metal, statements with respect to the properties of the oxide vary with different authors. In a very recent article, De Boer[13] gives the following data:

> The density of the monoclinic form is 5.56; that of the tetragonal form is 6.10.
>
> The melting point is very high, 2,950° ± 20° K. The oxide is a good insulator for heat and electricity. The coefficient of expansion is about 8.10^{-7}, only a little higher than that of silica, at least when the oxide has been melted before. Corresponding to the change in modification, there is a contraction at about 1,000° C. Heated in an oxyhydrogen flame, the oxide gives a brilliant white light.
>
> The hardness of well-ignited zirconium dioxide (a hard, white powder) lies between 6 and 7.
>
> From a chemical point of view, zirconium dioxide is a very stable product. This can be seen from the exceptionally high heat of formation, which is 264.0 K. cal. per mol. (out of metal and oxygen). When it is ignited, it is practically insoluble in all acids, except hydrofluoric acid, which dissolves it with a very violent reaction, provided the oxide was not ignited at too high a temperature. It is attacked by melted alkali bisulphate, caustic alkalies, or alkali carbonates. Heated with carbon, it will yield the carbide ZrC. Heated with carbon in a chlorine stream, it gives the tetrachloride.

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11 - Marden, J. W., and Rich, M. N., Work cited, p. 20.
12 - De Boer, J. H., Zirconium: Foote-Prints on the Rare Metals and Unusual Ores, vol. 3, No. 2, Foote Mineral Co. (Inc.), Philadelphia, Pa., 1930, pp. 6-7.
13 - De Boer, J. W., Work cited, pp. 7-8.

For a description of the nature and properties of the other compounds of zirconium, the reader is referred to the works of Venable,[14] of Marden and Rich,[15] and of De Boer,[16] quoted in preceding paragraphs.

HISTORY OF THE ELEMENT

The following chronology gives the names of scientists connected with the discovery of the element and its history step by step.

1789--Klaproth, while studying samples of the mineral zircon from Ceylon, found a large amount of an oxide resembling alumina closely but differing from it in being insoluble in alkalies. To this oxide the name "zirconium" was given, a name that is probably a corruption of "jargon," a Persian word meaning gold-colored.

1818--Berzelius pointed out the resemblance of the oxide to alumina and gave it the formula Zr_2O_3.

1824--Berzelius prepared the metal as an iron-gray powder by reducing potassium fluozirconate with potassium.

1845--Svanberg announced his belief that the oxide is a mixture of at least three earths; and he reported the new earth noria.

1857--Deville and Troost determined the vapor density of the chloride, that the element is quadrivalent, and that the formula ZrO_2 must be assigned to the oxide.

1859--Rose showed that the formula ZrO_2 was in accord with the isomorphism of rutile (TiO_2) and zircon (ZrO_2,SiO_2).

1860--Marginac observed the isomorphism between the fluozirconates of zinc and nickel and the fluosilicates, fluotitinates, and fluostannates of these metals.

1864--Nylander claimed that he had separated the new oxides from zirconia.

1865--Troost claimed that he had prepared the crystalline metal.

1869--Sorby and Forbes announced the discovery of jargonia, a new oxide. Church found the new element nigrium.

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14 - Venable, Francis P., Work cited, pp. 35-125.
15 - Marden, J. W., and Rich, M. N., Work cited, pp. 13-19.
16 - De Boer, J. H., Zirconium: Foote-Prints on the Rare Metals and Unusual Ores, vol. 3, No. 2, Foote Mineral Co. (Inc.), Philadelphia, Pa., 1930, pp. 6-13.

1901--Hofman and Prandtl claimed to have separated euxenerde from the zirconia of euxenite.

1923--Coster and Hevesy announced that examination of the X-ray spectra of the commoner zirconium minerals disclosed the presence, often in considerable quantity, of element 72 (for which they proposed the name "hafnium"), which remains closely associated with zirconium in the extraction of that element and is not easily separated from it.

USES

The first recorded use of zirconium or its compounds was the employment of the natural silicate as gem material. Zircon jewelry had been sold in very small amounts for many years, and recently zircon stones have become much more popular, largely owing to the introduction of the so-called "blue variety," named "Starlite" by Dr. Kunz.

The widest application of zirconium at the present time, however, is the use of the oxide as refractory material; and the second most extensive use, it is claimed, is that of the oxide as an opacifying agent, especially in enamel ware, as a substitute for tin oxide. Likewise, the use of the oxide as an opacifier in lacquers and automobile enamels seems to be of increasing importance.

Growing utilization of the pure metal is indicated by the decided drop in prices in 1929. Also more extended use of the metal alloys in steel is signified by a definite increase, in 1929, in the imports into the United States of steel-hardening ore.

Refractory Uses

Both zirconia (the pure zirconium oxide) and zirkite (the native Brazilian oxide or a semimanufactured form carrying 75 to 85 per cent of oxide) are employed as refractory material; however, as the pure oxide is too costly for large-scale industrial use, zirkite is generally substituted.

Because of its high fusing point (about 3,000° C.), zirconia is one of the most refractory oxides known. It does not melt below 2,560° C. even when it contains approximately 1.25 per cent of silica and ferric oxide. The pure fused material has very low thermal conductivity and a very low coefficient of expansion (8.4 times 10^{-7}), which compares favorably with that of carborundum (6.58 times 10^{-6}) or alundum (7.1 times 10^{-6}). Zirconia withstands sudden changes of temperature remarkably well, and it is chemically inert, being highly resistant to acids, fused quartz, and molten glass, and to a large extent even to fused alkalies. Under certain conditions, however, it has a tendency to be changed at high temperatures into a nitride or a carbide.[17]

17 - Rodd, E. H., Zirconia as a Refractory: Mineral Foote-Notes, July-Aug., 1918, Foote Mineral Co., Philadelphia, Pa., 1918, p. 7.

Brick

Improved methods of working zirconia into bricks and the addition of minimum quantities of magnesia or lime to obviate their former tendency to crack have greatly increased the use of zirconia or zirkite bricks. The various standard shapes are finding wide application in the United States, especially for lining electric furnaces.

With respect to the qualities of this form of refractory material, La Coultre and Carl[18] stated that the softening temperature for zirconia brick is 900° C. higher than that of fire clay and 600° higher than that of carborundum. The melting point of the brick, 2,600° C., is the same as its softening temperature. Zirconia bricks are inert to chemical influence; they are not attacked by hydrofluoric acid or by bisulphates; they are almost twice as brittle as fire clay or silica bricks and six times as brittle as alumina bricks; their compressive strength is nearly double that of fire clay; their tensile strength is much greater than that of other refractories; and their wear in furnaces is much less than that of silica bricks. The greater wearing quality of the bricks insures longer life and reduction in repair charges for a furnace, compensating for the higher initial cost of the bricks.

With respect to manufacturing processes for zirkite bricks, Meyer says[19] that the American method of manufacture consists essentially:

> ------In first passing the ore through a crusher and grinding it to about 60-mesh in a dry pan, removing all particles running finer than 100-mesh by passing it over inclined screens and bonding the resulting product with zirkite cement. About 50 per cent of 60-mesh zirkite and 50 per cent of zirkite cement constitute the refractory mass. This is made into a stiff mud with water and moulded in the same fashion as silica brick. "Green" zirkite bricks have to be dried very slowly, as otherwise they develop air cracks. Furthermore, great care should be exercised in setting them, as they are very brittle in the unburnt state. The percentage of loss is rather high, but the warpage and cracked bats can be used over again after regrinding in the dry pan.

Cement

The Foote Mineral Co. offers "Zirkite" cement in two grades, both of high-melting point, low coefficient of expansion, low thermal conductivity, and high resistance to corrosive slag and gases. One grade, the plastic grade (in the form of dry powder), carries 75 to 80 per cent of pure zirconium oxide,

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18 - La Coultre, _____, and Carl, _____, Zirconia Bricks: Chem. Age, vol., 17, London, Oct. 1, 1927, pp. 28-29 (of the Monthly Metall. sec.).

19 - Meyer, H. C., Zirconia; Its Occurrence and Application: Trans. Ceramic Soc., vol. 18, 1918-19, Stoke-on-Trent, 1919, p. 273.

has a softening point of 1,750 to 1,800°C., and sets at room temperature to a hard mass within a few hours. It is especially suitable for rammed-in linings or patchings. The nonplastic grade, which lends itself especially to "laying up zirkite bricks," does not harden at room temperature but at a temperature of about 500° C.

Another product, trade-marked "Zirkonalba," a chemically pure oxide, is used as a cement and also in the manufacture of articles subjected to exceptionally high and rapid fluctuations of temperature or to the corrosive action of slags and furnace gases. A melting point of 2,500 to 3,000° C. (4,503 to 5,432° F.) is claimed for Zirkonalba.[20]

Laboratory Ware

With respect to the manufacture of highly refractory utensils for laboratory equipment and similar ware, Meyer states[21] that zirkite filtering crucibles, muffles, combustion tubes, combustion boats, pyrometer tubes, and Kipp generators, with replaceable units, are now on the market at prices that compare favorably with like articles of porcelain or fused silica; that combustion tubes have run in steel-testing laboratories on carbon determinations for as long a period as three months, in constant use day and night; that because of the composition of these tubes they are not attacked by basic substances, do not devitrify, and are gas-tight up to temperatures of 1,000° C.

According to the Bureau of Standards,[21] zirconia crucibles (cast and fired as described below) have proved to be satisfactory for melting platinum (melting point, 1,755° C.) and rhodium (melting point, 1,950° C.) in air in high-frequency furnaces. These crucibles, which are best when made with an outer supporting crucible of alundum or porcelain, with the annular space packed with dry, purified zirconia, as they do not withstand sudden temperatures as well as could be desired, have been successfully used for making laboratory melts of platinum and platinum-rhodium alloys up to 300 or 350 grams in weight. The process of manufacture of this type of crucible is as follows:

> Commercial, electrically sintered and milled zirconia is washed three or four times, or until the washings are free from iron, in hot 1 : 1 hydrochloric acid. The acid-washed and dried zirconia is then roasted in an electric muffle furnace at 800 to 900° in porcelain evaporating dishes, with free access of air and with frequent stirring in order to burn out the carbon or carbides.
>
> The purified zirconia is then prepared as a casting "slip" by grinding with water and a small amount of china clay in a porcelain

20 - Mineral Foote-Notes, Zirconia: Vol. 3, No. 1, Foote Mineral Co., Philadelphia, Pa., 1930, p. 62.
21 - Meyer, H. C., Work cited, pp. 274-275.
22 - Jordan, Louis, Peterson, A. A., and Phelps, L. H., Refractories for Melting Pure Metals; Iron, Nickel, Platinum: Trans. Am. Electrochem. Soc., vol. 50, 1927, pp. 162-164.

ball mill with flint pebbles. The ball-mill charge is made in the proportions of 1,000 grams of purified zirconia, 40 grams of china clay, and 500 cubic centimeters of water. After grinding for approximately 8 hours, this slip is in a suitable condition for casting in plaster of Paris molds.

The cast zirconia crucibles are dried, first in air and then in an oven at 120 to 150° C., and are then fired to 1,700 to 1,800° C. in an Arsem furnace inside a covered protecting crucible of zirconium silicate.

Opacifying Uses

As an opacifier in enamels, as well as a clouding agent in glass, zirconia is an excellent substitute for stannic oxide, antimony oxide, and others. As a result of investigations in Germany, a number of proprietary compounds of zirconium oxide mixed with certain fluxes have been placed on the market, and many foreign patents have been issued covering the use of the oxide in white enamels for steel and cast-iron utensils.

The pure zirconium oxide is, it is claimed, superior to stannic oxide in that it is less easily reduced, is less volatile, and produces an enamel that is more resistant to vegetable and fruit acids. One authority maintains that zirconium compounds are more dependable than other substitutes for stannic acid in the opaquing of white glazes, 5 per cent of the oxide producing a good white glaze, whereas other authorities state that zirconium oxide has less covering power than stannic oxide, and that it is very difficult to obtain an iron-free product.

Minor Uses of the Oxide

The incandescence of zirconium oxide when heated was early taken advantage of in mantles, glowers, and lights. It first replaced the calcium oxide cylinders in the Drummond light; zirconia rods, heated to incandescence, were tried out for lighting Paris streets; an incandescent mantle was made that was composed chiefly of zirconia, which was soon replaced by thoria, however; in the Nernst light, small rods or glowers of zirconium, magnesium, and yttrium oxides were heated by an auxiliary device to a temperature of about 700° C., at which point they became conductors of electricity and through their resistance to the passage of current became incandescent. Except in the Bleriot light, which has been adopted somewhat extensively abroad, the quantity of zirconia employed in these connections is small.

Zirconia, as "Kontrastin," has been substituted for bismuthyl nitrate as a lining for the stomach in connection with X-ray photographs. Because of its hardness, its stability toward chemical reagents, and its volume, zirconia has been suggested for polishing and toilet powders. Small additions of zirconia to fused silica ware prevent devitrification. Finely divided oxide incorporated with rubber before its vulcanization increases its toughness and accelerates the vulcanizing process. Zirconium oxide produces a nonpoisonous, nondiscoloring white paint; it is used in ink, water-color paints, and like products.

Uses of Other Compounds

The carbide has been patented as a filament for incandescent lamps. It has been suggested as an abrasive, an experimental cement wheel giving good results with mother-of-pearl. It can not be used, however, in ceramic wheels, as it reacts with silicates at high temperatures.

The nitrate has been employed as a food preservative and in the lighting fluids used in the manufacture of incandescent mantles.

The acetate has been used in the place of stannic salts for the weighting of silk, a 50 per cent weighting being feasible.

Zirconium hydroxide has been considered for use in the purification of water. Zirconium compounds are used as mordants in the dyeing industry and in the preparation of lac dyes. Zirconyl tannate may find use as a substitute for sodium tungstate or stannate in rendering cloth noninflammable. The tetrachloride has been suggested as a chlorinating agent.

Uses of the Metal

Probably the chief commercial use of the metal to-day is in flashlight powders. Usually a mixture containing 60 per cent of zirconium and 40 per cent of magnesium is used, with barium nitrate as a primer. Another flashlight device consists essentially of aluminum leaf covered with zirconium-metal paste, which is set off when connected with an electric lighting circuit.

Zirconium metal is employed in radio tubes, in which it acts as a "getter."

Investigators have succeeded in producing a malleable zirconium as a substitute for platinum in the chemical laboratory. The metal is suitable for use in the manufacture of scientific apparatus and most delicate instruments and is said to be an ideal material in dentistry, because of its malleability, ductility, noncorrosiveness, and imperviousness to the action of gases.

The metal is of peculiar value for use on spark plugs, thermocouples in pyrometers, and other instruments for measuring heat, because of its high fusing temperature.

Research work has been carried on with reference to the employment of zirconium for the lining of cylinders of internal-combustion engines and for the manufacture of pistons.

The fact that zirconium has a pronounced electronic valve action is noteworthy, although zirconium electrodes may not, when available, be of great value in connection with valve cells. A new resistor, which has simplified the manipulation of currents used for impulse tests or measuring, has been developed from a so-called "zircon mixture." These zircon rods are particularly suitable for field investigations of lightning disturbances on transmission lines.[23]

23 - Beck, Edward, A New Resistor for Use in Measurements of Impulse Voltages and Currents: Electrical Jour., vol. 25, No. 3, Mar., 1928, pp. 157-158.

ZIRCONIUM ALLOYS

A few hundred tons of zirconium ore is used annually in the production of zirconium alloys. These alloys, which are produced in electric furnaces, are employed principally in the production of steel castings. The substitution of zirconium-ferrosilicon for the ordinary ferrosilicon makes possible a sounder and cleaner product, which is desirable in steels for castings, forged products, and rolled shapes that must be subjected to repeated and violent shocks. Silicon-zirconium is preferred for removing objectionable gases and nonmetallic impurities in the finer alloy steels.

Although it seems to be generally agreed that zirconium gives to alloyed metals increased tensile strength, toughness, and some degree of malleability, accurate information concerning the properties of zirconium alloys is rather meager, and conflicting opinions are expressed in published statements in the literature. Until very recently the claim was made that zirconium did not alloy with iron, but now ferro-zirconium, zirconium-ferrosilicon, and silicon-zirconium are on the market and are strongly recommended for use in steel, as well as in nickel alloys. Zirconium does not alloy with tin, lead, or metals of the alkali or alkaline-earth groups. Cobalt, aluminum, and magnesium alloys have been placed on the market, and silver alloys have been prepared.

The following summary of the value of zirconium in steel is from the catalog of the manufacturer of zirconium alloys:[24]

> When added to steel, zirconium performs three important functions. By its powerful deoxidizing action it vigorously reduces metallic oxides and scavenges nonmetallic inclusions, but it does not tend to remain in the steel in the form of an oxide, as does aluminum. By reason of the cleansing it thus receives, the steel becomes homogeneous both as to composition and structure. Secondly, zirconium combines chemically with nitrogen dissolved in the steel, forming a nitride of which by far the greater part enters the slag; the few nitride crystals remaining in the steel are too minute to have any effect on its mechanical properties. Finally, zirconium combines chemically with sulphur abstracted from the iron, the resulting sulphide being malleable like that of manganese; moreover the hot-shortness of high-sulphur steel can be diminished through the agency of zirconium to a degree not attainable by manganese, but for economic reasons it is not suggested that zirconium should displace manganese entirely in the treatment of high-sulphur steels.
>
> A zirconium-treated carbon steel is therefore characterized by: freedom from inclusions, which would act as nuclei for shock and fatigue failures; uniformity of grain, which assists in the effectiveness of heat treatment; excellent hot-working properties; and, when suitably heat-treated, tensile properties closely approaching those of special alloy steels, ductility being especially favored by zirconium treatment.

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24 - Electro-Metallurgical Sales Corporation, Carbide and Carbon Bldg., 30 East 42d Street, New York, N. Y.

The alloy generally used is zirconium-ferrosilicon, which contains 13 or 14 per cent of zirconium and 45 per cent of silicon, and which costs (Jan., 1931) $103.50 a gross ton in carload quantities. Where a large excess of silicon would be undesirable, silicon-zirconium, containing 35 to 40 per cent of zirconium, is used. Its cost, however, is (Jan., 1931) 18 cents a pound.

"Cooperite" is the name given to a patented zirconium-nickel alloy for use in the manufacture of edge tools of all kinds, especially machine tools for milling cutters and cast tools for lathe and plane. When the alloy contains 2 to 10 per cent of zirconium, the remainder being nickel, it is suitable for knives, razors, and like articles, which remain bright and clean under the action of acids such as that of the lemon. Since it can be worked at a red heat, it is applicable for electrical toasters, irons, and other domestic utensils. When the alloy contains 25 to 30 per cent of zirconium, it is suitable for high-speed cutting tools, though some other metal, such as molybdenum, is sometimes added to increase the melting point or the hardness of the metal.

A zirconium-nickel alloy patented by Cooper (composition: Al, 8.36; Si, 3.80; Zr, 6.84; and Ni, 81) was produced as follows:

> Sixty-six pounds of nickel oxide and 36 pounds of Brazilian zirkite were ground to pass a 200-mesh sieve and were thoroughly mixed with 32 pounds of 200-mesh magnesia or alumina. The charge was ignited with magnesium ribbon. The alloys were tapped into molds, remelted, and further purified to give the desired percentage composition.

ZIRCONIUM-BEARING MINERALS

The better-known zirconium-bearing minerals are tabulated by Watson and Hess as follows:[25]

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25 - Watson, Thomas, and Hess, Frank L., Properties, Occurrence and Uses of Zircon: Min. Eng. World, vol. 37, July 6 to Dec. 28, 1912, p. 951.

Mineral	Composition	Inclosing Rock	ZrO_2 Per cent
Oxide:			
Baddeleyite (Brazilite)	ZrO_2	Igneous rocks deficient in silica, and in gravels derived from them	100
Zirconates:			
Zirkelite	$(Ca,Fe)O.2(Zr,Ti,Th)O_2$	Magnetite-pyroxenite (jacupirangite)	52.89
Polymignite	$5RTiO_3.5ZrO_3.R(Cb,Ta)_2O_6$	Elaeolite syenite	29.71
Silicates:			
Zircon	$ZrSiO_4$	Variable, described below	67.2
Cyrtolite	Some cyrtolite is probably hydrated zircon	Granite, pegmatite	66.93
Catapleiite	$H_2(Na_2,Ca)(Zr(OH)_2)(SiO_3)_3$	Elaeolite syenite	28.8
Elpidite	$H_6Na_2ZrSi_6O_{18}$	Elaeolite syenite (?)	20.48
Eudialyte (Eucolite)	$Na_{13}(Ca,Fe)_6.Cl(Si,Zr)_{20}O_{52}$	Elaeolite syenite	16.88
Hainite	Related to låvenite, wohlerite, etc.	Phonolite	Unknown
Hjörtdahlite	$4Ca(Si,Zr)O_3.Na_2ZrO_2F_2$	Elaeolite syenite	21.48
Låvenite	$(Na_4,Ca_2,Mn_2,Zr)((Si,Zr)O_3)_2$	Elaeolite- or augite syenite	31.65
Lorenzenite	$Na_2Si_2(Ti,Zr)_2O_9$	Pegmatite	11.92
Rosenbuschite (Zircon pectolith)	$Na_2Ca_3((Si,Zr,Ti)O_3)_4$	Elaeolite syenite	20.10
Wöhlerite	$(Na_2,Ca)(Si,Zr)O_3.RCb_2O_6$	Zircon syenite	22.72

The orthosilicate zircon and the oxide baddeleyite are practically the only zirconium minerals of commercial importance, but certain of the others, notably eudialyte and the altered zircons, alvite, cyrtolite, and malacon, have recently become of scientific interest, in that they have been proved to be among the richer hafnium-bearing minerals.

Zircon

Zircon, the most widely distributed and the most abundant zirconium mineral, is an orthosilicate, having the formula $ZrSiO_4$, and containing (when pure) 67.2 per cent of zirconia and 32.8 per cent of silica. Frequently iron is present in small quantities. Zircon occurs in crystals and in formless grains. The crystals are tetragonal, commonly in square prisms, sometimes elongated, having pyramidal terminations. The crystals sometimes show geniculated twinning. Large perfect crystals are rare and, when found, are classed as gems.

Zircon is harder than quartz (hardness, 7.5) and fairly heavy (sp. gr. 4.7). The molecular weight is 183, and the molecular volume is 38.7. It is infusible, and it is insoluble in most acids but is attacked when in powdered form by concentrated sulphuric acid. It is brittle, has imperfect cleavage, conchoidal fracture, and adamantine luster. It is generally opaque, and the color is usually some shade of brown, although it may be clear yellow, gray, green, red, or some other color, or (rarely) colorless.

Mode of Occurrence

Zircon occurs (1) as a product of magmatic segregation, (2) in pegmatite dikes, (3) occasionally as a product of contact metamorphism, and (4) as a product of dynamo-regional metamorphism.[26] Classes 1 and 2 constitute the principal sources.

As an accessory constitutent, zircon is found in nearly all classes of plutonic and volcanic rocks. It is a very common constituent of granites and syenites and occurs in some diorites and gabbros (among plutonic igneous rocks) and in quartz-porphyry, trachyte, phonolite, tephrite, dolerite, diabase, and basalt (among volcanic igneous rocks).

Zircon is a frequent associate in beds of sand and gravel, and hence, it is found in various sedimentary rocks, including even limestones. On account of its density, it is one of the minerals left behind by transporting agencies; so that it is a natural concentrate in river and sea sands and gravel. Its high degree of hardness enables it to resist wear as it is being carried by running water, and its great durability enables it to resist weathering.

Zircon, occurring in almost all classes of metamorphic crystalline rocks, is found especially in gneisses rich in feldspar that are of probable igneous origin and also in different types of crystalline schists, particularly the hornblendic, chloritic, and micaceous groups. It is usually found also in quartzites.

Zircon has been noted as an accessory mineral in iron-ore deposits, particularly magnetite deposits.

Identification

According to Zealley,[27]

> *****zircon is easily recognized by its physical properties and by the following simple blowpipe test: Fusing the powdered mineral with sodium carbonate, dissolving the melt in dilute hydrochloric acid, and adding turmeric paper. The presence of zirconia is revealed by the paper's turning to an orange color.

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26 - Watson, Thomas L., and Hess, Frank L., Work cited, p. 952.

27 - Zealley, A. E. V., Zirconium Ore: Rept. Rhodesian Resources Committee, 1921, Bulawayo, 1921, p. 130.

Baddeleyite

Baddeleyite, much less widely distributed than zircon, is the dioxide of zirconium (ZrO_2). This mineral has been found in quantity in Brazil only; however, it has been identified in Ceylon, Sweden, Italy, and Montana. In fibrous, botryoidal, or columnar forms, the oxide is termed "brazilite," whereas the name "baddeleyite" refers to the mineral when it occurs in distinct crystals. Jacupirangite is the name applied to a crystalline variety of baddeleyite, named from Jacupirangi (or Jacupiranga), Brazil. Baddeleyite is not quite so hard as zircon (hardness, 6.5) but is heavier (specific gravity 5.5 to 6.0). The crystals are colorless, yellow, brown, or black.

ZIRCONIUM ORES

As previously noted, zircon and baddeleyite are the only zirconium-bearing minerals occurring in sufficient abundance to form ore deposits. The following table is a convenient classification of commercial minerals.

Commercial classification of zirconium ores[1]

Name	Formula	Per cent ZrO_2
Baddeleyite (distinct crystals)	ZrO_2	96.5-98.9
Brazilite	ZrO_2	71.93
Zircon	$ZrO_2.SiO_2$	67
Zirconia ore: 1. Favas (alluvial pebbles)	ZrO_2	59-92.4
2. "Zirkite" (mixture) (Brazilite	ZrO_2	71.93
(Zircon	$ZrO_2.SiO_2$	67
(Unnamed		
(Zr silicate		
(Orvillite (?)		

1 Lee, O. Ivan, The Mineralogy of Hafnium: Chem. Rev., vol. 5, No. 19, Am. Chem. Soc., 1928, p. 18.

In the trade, the name "baddeleyite" seems to be confined to the variety in distinct crystals and the name "brazilite" to the fibrous, botryoidal, or columnar form. The trade name "Zirkite" is applied to a mixture of baddeleyite or brazilite, zircon, and a supposed new and unnamed silicate of zirconium. Zirkite contains from 70 to 94 per cent of zirconium oxide. The following variations in the analyses of commercial ores were reported by Schaller:[28]

28 - Schaller, Waldemar T., Zirconium and Rare-Earth Minerals: Mineral Resources of the United States, 1916, pt. 2, U. S. Geol. Survey, 1919, p. 377.

Analyses of zirkite

	1	2	3	4	5	6	7	8	9	10	11
ZrO_2	93.18	81.75	86.57	85.93	82.00	85.01	71.88	94.12	88.40	74.48	68.93
SiO_2	1.94	15.49	2.50	9.35	11.38	9.63	25.31	2.41	5.89	14.08	26.30
TiO_2	.69	.50	1.43	1.84	.36	1.52	.63	.98	3.12	1.35	.60
Fe_2O_3	2.76	1.06	5.29	1.93	2.08	3.57	.43	3.22	4.07	10.26	3.59
Al_2O_3	.64	.85	1.00	.36	.62	--	.15	--	--	--	--
MnO	Trace	Trace	--	--	--	--	--	--	--	--	--
H_2)	.47	.63	3.32	1.56	3.35	--	1.56	--	--	--	.80
	99.68	100.28	100.11	100.97	99.79	99.73	99.96	100.73	101.48	100.17	100.22

1. Slate-gray fava (waterworn pebble resembling a bean). Specific gravity 5.245.
2. Light-brown fava. Specific gravity 4.850.

3-6. Hard lump ore.

7. Gray porous ore.
8. Glassy variety.
9. Stony variety.
10. Pebbles.
11. Commercial variety.

Analyses of baddeleyite

	1	2	3		1	2	3
ZrO_2	97.19	98.90	96.52	MnO	Trace	--	--
SiO_2	.48	.19	.70	MgO	--	--	.10
TiO_2	.48	--	--	Alkalies	--	--	.42
Fe_2O_3	.92	.82	.41	H_2O	.38	.28	.39
Al_2O_3	.40	--	.43				
CaO	--	.06	.55		99.85	100.25	99.52

1. Botryoidal, from Brazil. Specific gravity 5.533.
2. From Ceylon. Specific gravity 5.72 to 6.025.
3. From Jacupirangi, Brazil. Specific gravity 5.006.

GEOGRAPHICAL OCCURRENCES

Zirconium minerals are widely distributed, but commercial deposits are few.

In the United States, deposits of beach sands have been worked, mostly in conjunction with monazite and titanium (ilmenite), the only one reporting commercial production being that at Mineral City, Jacksonville Beach (formerly Pablo Beach), where Buckman & Pritchard, Inc. (later a subsidiary of the National Lead Co.) operated from 1918 to 1929. More or less extensive deposits of such sands are reported to exist elsewhere along the Atlantic coast; and in the black sands found along the Pacific coast (principally in northwestern California, in Oregon, and in Washington) zircon ranks fifth in order of frequency among the more than 37 minerals identified therein.

The only other deposit in the United States that has been systematically mined is one of zircon (in pegmatite and pegmatized gneiss) in Henderson County, N. C., near Tuxedo (formerly Zirconia), where mining began on a very small scale as early as 1869; no recorded shipment, however, was made until 1888. Intermittent production continued until 1911, was renewed in 1902, and probably continued until 1916, although accurate figures are not available with respect to output.

Deposits of zircon (in sandstone) at Ashland, Va., and (in pegmatite) at Anderson, S. C., have been mentioned as promising sources of the mineral.

Most of the world's production of zirconium-bearing minerals, however, has come from Brazil, where the only known commercial deposit of baddeleyite (the native oxide) exists, in the States of Sao Paulo and Minas Geraes, in the so-called "Caldas Region." Brazil has important deposits of zircon (the silicate) also, in the States of Bahia, Espirito Santo, and Rio Janeiro, where zircon is associated with the monazite.

Another important source of zirconium is the beach-sand deposit in the Travancore State, India, at the extreme end of the Hindustan Peninsula.

Workable deposits exist also in Madagascar, in the vicinity of Beforona and west of Fianarantsoa, from which regular production has been reported, in relatively small amounts, however.

MINING AND CONCENTRATION

Because of the hardness of the ore, the primitive methods of the emery miners of Naxos were used in the deposits of the Caldas region (States of Minas Geraes and Sao Paulo, Brazil) in 1916, according to Meyer.[29] A fire was built against the exposed face of the ore and kept burning for several hours, and then water was thrown upon the ore to produce fracturing of the mass. According to Cameron,[30] however, dynamite is now being used. In some of the deposits, the ore occurs as lumps and granules in a reddish clay matrix and is mined by open-cut methods. The clay, which readily dries when exposed to the tropical sun, can be separated from the zirconia by a coarse screen.[31] From these deposits the ore is carried about 20 kilometers by carts, loaded on railway cars, and shipped a distance of about 230 miles. The beach deposits, especially those near Victoria (State of Espirito Santo), mined as a by-product of the monazite industry, are surface-mined or quasi surface-mined, there being an overburden of approximately 2 meters of ordinary quartz sand and underlying layers of ilmenite, zircon, and monazite in the proportions of 35 : 25 : 18, respectively. Electromagnetic separators extract first the ilmenite, then the zircon, and finally the monazite.[32]

29 - Meyer, H. C., Brazilian Zirkite Deposits: Monthly Prices for November, Foote Mineral Co., 1916, pp. 29-31; Schaller, Waldemar T., Zirconium and Rare-Earth Minerals: Reprint from Mineral Foote-Notes, vol. 2, No. 3, Foote Mineral Co. (Inc.), Philadelphia, Pa., Mar., 1918, pp. 6-8.

30 - Cameron, C. R., American consul general, Sao Paulo, Brazil: Trade letter, May 28, 1930, Bur. For. and Dom. Com. file.

31 - Meyer, H. C., Work cited, p. 6.

32 - Cameron, C. R., Work cited.

In the Florida beach deposits, the sand was loaded into trucks for transportation to the mill or, over part of the beach, was transported over a narrow-gage tramway. An attempt to mine by dredging was not successful, because the rich concentrates do not extend below water level.

In the concentration of zircon or monazite sand, the sand is usually passed over magnetic separators. According to Martens,[33] the Florida beach sands, worked by Buckman and Pritchard (Inc.), were treated first on shaking tables in a wet mill. The table concentrates contained about 20 per cent zircon, together with ilmenite (55 per cent), rutile (6 per cent), monazite (2 per cent), various heavy green silicates (14 per cent), and a little quartz (3 per cent). Dry magnetic treatment removed first the ilmenite and also the so-called "greenspar" (mostly staurolite and epidote), together with most of the monazite, leaving tailings consisting principally of zircon and rutile. These tailings were run over a wet table to eliminate some still remaining quartz and silicate minerals, and the zircon and rutile were separated on an electrostatic machine.

PREPARATION OF THE OXIDE

The oxide is commercially the most important of the various compounds of zirconium. For a complete discussion of the other compounds the reader is referred to the work of Venable,[34] from which the following quotation concerning zirconia (ZrO_2) is taken:

> Preparation of oxide from, or purification of, native zirconia.- For many purposes native zirconia can be used without further treatment. Such uses, for instance, are as a refractory and for furnace linings. However, for the preparation of enamels, salts, and for other uses requiring purity of materials the impurities must be separated. By fusion with an alkali hydrogen sulphate and leaching with water acidulated with sulphuric acid, a solution of the sulphate is obtained. Much of the iron present may be removed by using a small amount of water for the first washing and rejecting this. The solution of the sulphate yields the hydroxide on the addition of ammonia solution. This is dissolved in dilute hydrochloric acid and the zirconyl chloride purified by recrystallization from concentrated hydrochloric acid. On ignition of the chloride zirconia is left, but small amounts of chlorine are persistently retained.
>
> A more economical method has been patented for large-scale production. The ore is heated with excess of lime and an amount of carbon insufficient for the reduction of the lime. Calcium carbide may be used in the place of the carbon. The product is treated with hydrochloric acid, the silica removed, and the zirconyl chloride then purified.

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33 - Martens, H. C., Beach Deposits of Ilmenite, Zircon and Rutile in Florida: Nineteenth Ann. Rept., Florida State Geol. Survey, 1926-27, The State Geological Survey, 1928, pp. 136-140.

34 - Venable, Francis P., Work cited, pp. 36-37.

Preparation of zirconia from zircons.- The most direct method is by subjecting powdered zircons to the high temperature of the electric furnace. The silica and oxides of iron and certain other metals are volatilized, and the zirconia is left in quite a pure condition.

Another commercial method for the preparation of zirconia in a relatively pure state from the native oxide is suggested by Bradford:[35]

1. Fusion of the finely powdered ore with niter cake (low in iron).

2. Leaching of the mass and separation of the silica by filtering through asbestos mat or other suitable medium.

3. Precipitation of zirconia by commercial sodium thiosulphate after nearly neutralizing the solution with sodic carbonate and precipitating at the boiling point.

4. Filtration of the precipitate, thoroughly washing in a small filter press to remove the solution containing all the iron salts.

5. Ignition of the predried cakes in the fire-clay pots.

By this method, which could be utilized in nitric acid works, an oxide could be produced containing only titania and a little alumina (which would not be harmful for most purposes) and a trace of iron.

Hopkins[36] discusses the extraction (by chemical treatment) of zirconium material from zircon and from baddeleyite, as follows:

> From zircon, extraction may be made by several methods: (1) fusion with acid potassium fluoride and extraction with hot water; potassium fluozirconate crystallizes on cooling; (2) fusion with potassium bisulphate and extraction with dilute boiling sulphuric acid; the basic sulphate is left undissolved; (3) heating with NaOH and NaF and extracting with water; the sparingly soluble sodium zirconate is dissolved in HCl, and on evaporation $ZrOCl_2$ separates; (4) fusion with Na_2CO_3 and extracting with water. The zirconate is dissolved in HCl or H_2SO_4.

35 - Bradford, Leopold, Zirconia; Its Possibilities in Metallurgy: Mineral Foote-Notes, vol. 2, No. 7, Foote Mineral Co., July-Aug., 1918, p. 3.

36 - Hopkins, B. Smith, Chemistry of the Rarer Elements: D. C. Heath & Co., New York, 1923, pp. 150-151.

The preparation of zirconium material from baddeleyite usually consists in the removal of iron, silica, and less commonly titanium. Fe_2O_3 is present in amounts varying from 0.82 per cent to 10.26 per cent, silica 0.19 to 26.30, and titania from a trace to 3.12 per cent. The method of treatment depends upon the purity desired in the product. (1) Boiling with strong HCl or H_2SO_4 removes most of the iron and titanium; ignition with HF and H_2SO_4 removes the silica. (2) For a product free from impurities, it is recommended that the ore be fused with Na_2CO_3 and $Na_2B_4O_7$. The melt is extracted with water and the zirconium salt crystallized out. (3) Fusion with NaOH in an iron crucible, followed by extraction with water and HCl, then precipitation of basic zirconium sulphate yield a pure product. (4) Fusion with $BaCO_3$ at 1400° for two hours gives barium zirconate, which may be dissolved in HCl and the solution evaporated to remove silica. Dissolve the zirconium in acid and allow $ZrOCl_2$ to crystallize. (5) By heating the ore in the electric arc with carbon the nonvolatile zirconium carbide is formed and is easily separated from the volatile silicon carbide. Aqua regia is used to dissolve the ZrC. (6) A method recommended for the large-scale purification of zirkite consists in mixing the ore with sufficient carbon to form silicon carbide, then heating in an arc furnace to a temperature above 2220°. This removes 90-95 per cent of the silicon, but no iron. Next the material is heated in the presence of chlorine or phosgene to remove iron.

As the zirconium compounds prepared directly from the ores invariably contain impurities (especially iron), further purification is necessary. Hopkins[37] presents the following methods:

1. Separation of iron and zirconium.--(1) $Na_2S_2O_3$, added to a boiling hot solution of zirconium salt which contains a little free acid, precipitates zirconia, thoria, and titania, but iron, aluminum, and the rare earths are not precipitated. (2) $(NH_4)_2S$ in the presence of tartaric acid precipitates iron but not zirconium. (3) Ether extracts $FeCl_3$ from a solution containing HCl, but not $ZrCl_4$. (4) Repeated crystallizations of the oxychlorides will separate iron and zirconium. (5) Vaporization of $FeCl_3$ at 200° C. (6) Phenyl hydrazine and sulphurous acid precipitate zirconium but not iron.

2. Separation of titanium and zirconium.--Zirconium may be separated from titanium by (1) boiling with dilute sulphuric and acetic acids, titanium being precipitated; (2) reducing titanium to the trivalent condition and precipitating potassium zirconium sulphate; (3) precipitating basic zirconium phosphate with hydrogen peroxide and sodium phosphate.

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37 - Hopkins, B. Smith, Work cited, pp. 151-152.

3. Separation of zirconia and silica.--Zirconia may be separated quantitatively from silica by fusion with Na_2CO_3; extract the melt with water, filter, evaporate with nitric acid, dehydrate the silica, and expel it with HF and H_2SO_4.

4. Separation of zirconium from titanium and tin.--Titanium and tin may be removed from zirconium by fusion with large excess of $KHSO_4$; dissolve in water and nitric acid, and ignite the SnO_2 residue. Carefully neutralize the solution and precipitate the zirconium with H_2O_2 and from the filtrate precipitate the titanium with NH_4OH.

PREPARATION OF THE METAL

The preparation of pure ductile zirconium is described by De Boer[38] and summarized in Industrial Chemistry and Engineering, as follows:

> There are three ways in which zirconium powder can be produced: (1) By the reduction of zirconium oxide with calcium; (2) by the reduction of zirconium tetrachloride with sodium; and (3) by the reduction of potassium fluozirconate with potassium or sodium.

> De Boer claims to have produced very pure coherent zirconium by depositing it on a heated tungsten filament. The zirconium powder prepared by one of the methods listed above is placed in an evacuated flask with a small amount of iodine. At about 500° C. the zirconium and iodine combine to form the tetraiodide, which vaporizes to the filament and is deposited thereon by thermal dissociation.

A method has been worked out by De Boer and Van Arkel for the preparation of pure zirconium and pure hafnium.[39]

Methods for the preparation of amorphous, crystalline, graphitic, and sintered or coherent (solid metal) zirconium are discussed in Bulletin 186 of the Bureau of Mines.[40] The authors thereof agree with Venable (p. 3 of this paper) that it is doubtful whether the crystalline and graphitic forms are pure metal.

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38 - De Boer, J. H., Zirconium: Ind. Eng. Chem., vol. 19, No. 11, Am. Chem. Soc., Nov., 1927, pp. 1256-1259.

39 - Tyler, Paul M., Hafnium: Information Circular, U. S. Bureau of Mines, to be published.

40 - Marden, J. W., and Rich, M. N., Investigations of zirconium with Especial Reference to the Metal and Oxide: Bull. 186, Bureau of Mines, 1921, pp. 35-37.

WORLD PRODUCTION

The zirconium industry is still in its infancy, and little information is available regarding the annual production of zirconium minerals. The workable deposits of zircon in the United States having ceased to produce, the output is confined to the native oxide, zirkite, mined in Brazil, and the zircon produced in Travancore (India), Brazil, and Madagascar.

Because of the incompleteness of available production figures, no attempt has been made to formulate a comprehensive world table. However, one table is introduced to show the output in the United States and Brazil for the years 1902 to 1919, inclusive, and another to show production in the United States, Brazil, India, and Madagascar for the years 1920 to 1929, inclusive.

Zirconium minerals produced, United States and Brazil, 1902-1919 1/

Year	United States		Brazil	
	Quantity (short tons)	Value	Quantity (short tons)	Value
1902	1	$1,380	12	$3,947
1903	1.5	570	7	1,947
1904	0.5	200	9	3,935
1905	4	1,600	18	5,506
1906	0.5	248	26	5,041
1907	(2)	46	38	8,756
1908	---	-----	275	15,151
1909	1	250	117	11,838
1910	---	-----	128	23,271
1911	1.6	802	45	16,169
1912	---	-----	43	14,772
1913	---	-----	1,119	54,767
1914	---	-----	237	14,903
1915	---	-----	8	2,915
1916	---	-----	104	16,647
1917	---	-----	532	175,018 Milreis 3/
1918	---	-----	2,360	507,019 Do. 3/
1919	---	-----	229	$18,478

1/ Figures of production for the United States are taken from Mineral Resources of the United States; those for Brazil are taken from various issues of Commercio Exterior do Brazil, Directoria de Estatistica.

2/ Two hundred and four pounds.

3/ Figures are given in milreis, as the rate of exchange for the year was not available.

Zirconium minerals produced, United States, Brazil, India, and Madagascar, 1920-1929

Year	United States 1/		Brazil 1/		India (Travancore) 2/		Madagascar 3/
	Quantity (short tons)	Value	Quantity (short tons)	Value	Quantity (short tons)	Value	Quantity (short tons)
1920	-----	-----	597	$25,034	--------	------	3.9
1921	-----	-----	134	6,095	--------	------	13,303
1922	10	4/ $1,000	366	16,844	--------	------	.746
1923	153	4/ 15,300	36	883	162.4	£1,160	27,593
1924	281	4/ 28,100	842	26,137	408.8	2,717	60,21
1925	624	4/ 62,400	137	4,142	645.12	4,608	4.191
1926	1,292	4/129,200	12	563	585.8	2,987	.274
1927	3,646	4/364,600	285	8,313	1,640.8	8,129	----
1928	(5)	(5)	913	31,048	957.5	4,267	----
1929	-----	-----	1,187	41,419	--------	------	24,759

1/ Figures for production for the United States are taken from various issues of Mineral Resources of the United States; those for Brazil from issues of Commercio Exterior do Brazil, Directoria de Estatistica.
2/ Figures for India are taken from records of the Geological Survey of India.
3/ Figures for Madagascar, which are the same as the export figures, are taken from Statistique Minière Productions et Exportations, Colonie de Madagascar et Dépendances, Service des Mines. Figures of value are not available.
4/ Estimated by the Bureau of Mines.
5/ Bureau of Mines not at liberty to publish figures.

DOMESTIC PRODUCTION

As early as 1869, zirconium minerals were mined in the United States to the extent of 1,000 pounds; and in 1883 a production of 26 tons was reported. There is no record of further production until 1902, when 1 ton, valued at $380, was mined. Although the mineral was known to exist in commercial quantities in Henderson County, N. C., the demand was too small to encourage any attempts at systematic mining. In 1903, men and children, for a certain price per pound, were washing zircon crystals (practically 100 per cent zircon) out of the soil or kaolinized gangue or breaking them from the harder feldspar in North Carolina. From 1903 to 1911, from ½ ton to 4 tons was produced each year (with the exception of 1908 and 1910), all in Henderson County, N. C. From 1912 to 1919, no minerals were reported in the United States, but in 1922 about 10 tons of zircon was shipped from Florida. Production from Florida increased steadily to 3,646 tons in 1927, and shipments continued into 1928, although the Bureau of Mines was not permitted to publish the figures. In 1929, the plant of Buckman and Pritchard, at Mineral City, Fla., was closed, and subsequently there has been no domestic output of record, although late in 1930 a small production from a new source was contemplated.

IMPORTS

In recent years the zirconium supply of the United States has been mainly imported, principally from Brazil and British India. Until 1918, the exact amount of imports was not known, as the negligible amounts of zirconium

ores that entered the country, being not separately classified, were included with monazite sand. The table, which follows, indicates imports from July, 1918, to 1929, inclusive.

Zirconium ores (steel-hardening) imported for consumption in the United States, 1918-1929

Year	Pounds	Value
1918--July to December 1/	3,216,659	$77,250
1919	11,023	332
1920	---------	-------
1921	131,016	3,003
1922	67,037	4,072
1923	---------	-------
1924	616,220	4,878
1925	---------	-------
1926	---------	-------
1927	4,488	182
1928	863,685	9,788
1929	2,689,120	35,416

1/ Not separately classified prior to July, 1918.

Imports of zirconium ores and zirconium alloys during 1929, by countries 1/

	Zirconium ore		Ferrozirconium, zirconium, and ferrosilicon	
1929	Pounds	Value	Pounds	Value
Argentina	433,440	$ 5,612	------	-----
Brazil	1,008,000	13,050	------	-----
Canada	------	----	20,221	$ 2,022
France	------	----	22,307	1,290
Germany	------	----	4,520	1,176
India-British	1,247,680	16,754	------	-----
	2,689,120	35,416	47,048	4,488

1/ From records of the Bureau of Foreign and Domestic Commerce, compiled by J. A. Dorsey, of the Bureau of Mines.

MARKETS AND PRICES

The market for zirconium ores is a narrow one, and published quotations covering a period of years are not available for tabulation.

The Foote Mineral Co. (Inc.) issues a price list covering zirkite (crude zirconium oxide), carrying a guaranteed minimum of 65 to 70 per cent of ZrO_2, the following analysis being typical:

	Per cent
Zirconium dioxide......	65.66
Silica................	21.06
Alumina...............	4.95
Titania...............	.92
Iron sesquioxide.......	4.57
Loss on ignition.......	3.01

In 1930, this company quoted $60 a ton for powdered zirkite, either coarse or fine, ground material (95 per cent through 200 mesh) being sold as "Zirkite Cement." On small lots higher prices were charged up to 4½ cents a pound on shipments of 200 to 800 pounds. Lump zirkite is 25 per cent less than powdered zirkite, but higher prices are asked for air-floated material, 99 per cent through 200 mesh. Zirkite fire brick, 75 to 80 per cent zirconium content, is quoted by the same company at $1.10 each for ordinary straights, wedges, or arches.

In 1929, low-grade (60 to 65 per cent) lump ore was obtainable at about $35 to $40 a ton.

Precipitated zirconia white is sold by the Foote Mineral Co., under the trade name "Zirconalba," and by the Titanium Alloy Manufacturing Co. as "Tam Opax." Zirconalba No. 1 (minimum 98 per cent zirconium dioxide), in lots of 100 to 400 pounds, was quoted in 1930 at 75 cents a pound. Zirconalba No. 2 (minimum 80 to 85 per cent zirconium dioxide) was offered at 38 cents a pound, in lots of 500 to 2,000 pounds. Tam Opax (zirconium oxide with no admixtures), according to a price list dated June 20, 1930, in 500-pound barrels, was priced at 30 cents a pound, f.o.b. Suspension Bridge, New York. Of the lower grades of zirconium oxide sold by the Foote Mineral Co., Zirkosil No. 1, minimum 60 per cent zirconium dioxide (ground 80 to 90 per cent through 200 mesh), in 20 to 30 ton lots, was quoted in 1930 at $80 a ton; Zirkosil No. 2, minimum 53 per cent zirconium dioxide (ground 80 to 90 per cent through 200 mesh), in 20 to 30 ton lots, was quoted at $70 a ton.

The above-recorded prices are more or less in line with those of former years. In 1918, imported Brazilian ores were quoted at $100 to $140 a ton, and prepared zirkite varied from 4½ to 7 cents a pound in carload lots. During 1920 and 1921 prices receded to between $50 and $90 for ore, and the price of zirkite bricks (standard shapes) dropped to 45 to 70 cents each. Ore prices have subsequently remained at $50 to $60 (rarely $80) a ton for high-grade lump. Prices of bricks were revised upward, beginning about 1925.

With respect to the oxide, the prices in former years varied so greatly from year to year and from grade to grade that comparisons are difficult. Early in 1920, the pure oxide (99 per cent grade) was quoted at $1.10 to $1.20, increasing by the end of 1921 to $1.25 a pound retail. In 1922, the range of price was from 75 cents to $1.25 for 99 per cent material, but by 1925 the minimum price had become the maximum, as the pure oxide was selling at 56 to 75 cents a pound, which latter price is the 1930 price for the 98 per cent Zirconalba, referred to in a preceding paragraph. Prices during 1926 and 1927 were as low as 35 cents a pound for what was termed "purified oxide."

During 1929 the price of zirconium metal dropped from $40 to $12 for the 98 to 99 per cent grade and to $8 for the 90 to 95 per cent grade.

High-grade ferrozirconium, powdered or crushed, was quoted by Foote Mineral Co. in 1930 at $1.20, in 200 to 300 pound lots. The following is a typical analysis.

	Per cent
Zirconium........	71
Titanium.........	4
Aluminum.........	4.5
Carbon...........	4.5
Manganese........	3.5
Silicon..........	12.5

From July 10, 1930, the Engineering and Mining Journal has been quoting prices on the cheaper alloys as follows: Siliconzirconium (47 to 52 per cent silicon and 35 to 40 per cent zirconium) at 18 to 21 cents a pound; zirconium-ferrosilicon (12 to 15 per cent zirconium and 39 to 43 per cent silicon) at $103.50 to $108.50 a gross ton.

LIST OF IMPORTERS, PRODUCERS, AND DEALERS

United States

Abel Brothers & Co., 16 Maiden Lane, New York, N. Y.
M. C. Cadwallader, 1317 La Veta Terrace, Los Angeles, Calif.
Charlotte Chemical Laboratories, Charlotte, N. C.
(Successors to Oliver Quartz Co.)
Crucible Steel Co. of America, 17 East 42d St., New York, N. Y.
Foote Mineral Co., 1609 Summer St., Philadelphia, Pa.
General Minerals Corporation, Suite 317-319, Mather Bldg., Washington, D.C.
Gordon & Rogers, 141 Broadway, New York, N. Y.
The Harshaw Chemical Co. of New York, 150 Nassau St., New York, N. Y.
(Successors to The Superfos Co., Inc.)
L. Heller & Son (Inc.), 15 West 47th St. , New York, N. Y.
O. Hemmel Co., 211 Fourth Ave., Pittsburgh, Pa.
Philip S. Hoyt, Las Tablas, N. Mex.
E. J. Lavino & Co., Bullitt Bldg., Philadelphia, Pa.
Levere Co., 94 Canal St., New York, N. Y.
Metal & Thermit Corporation, 120 Broadway, New York, N. Y.
Pennsylvania Salt Co., Juniper and Chestnut Sts., Philadelphia, Pa.
Philipp Bros. (Inc.), Woolworth Bldg., New York, N. Y.
August Rassweiter, 159 North State St., Chicago, Ill.
E. Schaaf Regelman, 220 Broadway, New York, N. Y.
Rogers, Brown & Crocker Bros. (Inc.), 21 East 40th St., New York, N. Y.
S. R. Scott & Co., 39 Broadway, New York, N. Y.
Suffern & Co., 135 Broadway, New York, N. Y.
M. L. Thomas, Crown Point Chemical Co. (Inc.), Crown Point, N. Y.
Union Carbide & Carbon Corporation, 30 East 42d St., New York, N. Y.
Varlacoid Chemical Co., 15 Moore St., New York, N. Y.
Welsbach Co., Gloucester, N. J.

Foreign Countries

A. J. Byington, Rua Alvares Penteado 4, Sao Paulo, Brazil
(owner of deposits).
Algernon Lewin Curtis, P. O. 61, Westmoor Laboratory, Chatteris, England.
A. C. de Freitas & Co., Hamburg, Germany.[41]
Elias Goebel & Son, Epterode, in Hessen Nassau, Germany
(manufacturers of crucibles of all kinds).
John Gordon, Rua Visc. Itaborahy 75, Rio de Janeiro, Brazil
(owner of monazite-sand deposits, which contain zircon).
W. C. Heraeus, Hanau, Germany.
Hopkin & Williams (Ltd.), 16 Cross St., Hatton Garden, London, England.
Mauricio Isralson, R. General Camara, 106 Sob., Rio de Janeiro, Brazil.
David McKnight, Rua Alamoda Barros 38, Sao Paulo, Brazil
(owner of deposits).
P. S. Nicholson & Co., Caixa 91, Rio de Janeiro, Brazil.
S. A. Casa Nicholson, Rua Theophilo Ottoni 45, Rio de Janeiro, Brazil
(owner of monazite-sand deposits, which contain zircon).
Luiz de Rezende & Co., Rua Ouidor, Rio de Janeiro, Brazil.
Charles Spitz, Rio de Janeiro, Brazil.[42]
Thorium (Ltd.), Ilford, Essex County, England
(associated with Hopkin and Williams).
Weserfeld, Dicke & Co., Barmen, Germany.

LIST OF POSSIBLE BUYERS

United States

Abel Bros. & Co. (Inc.), 16 Maiden Land, New York, N. Y.
Jerome Alexander, 50 East 41st St., New York, N. Y.
Eimer & Amend, 201-209 East 13th St., New York, N. Y.
The Exolon Co., Commercial Ave. and Erie R. R., Blasdell, N. Y.
Foote Mineral Co., 1609 Summer St., Philadelphia, Pa.
The Harshaw Chemical Co. of New York, 150 Nassau St., New York, N. Y.
(Successors to the Superfos Co., Inc.)
O. Hommel Co., 209-211 Fourth Ave., Pittsburgh, Pa. (Buyer of ore)
Juergens & Anderson Co., 53 East Washington St., Chicago, Ill.
Levere Co., 94 Canal St., New York, N. Y.
A. D. Mackay, 26 Cortlandt St., New York, N. Y.
F. E. Morse Co., 218 South Wabash Ave., Chicago, Ill.
National Sales Corporation, 31-35 East 13th St., Cincinnati, Ohio.
Norton Co., Worcester, Mass.
Philipp Bros. (Inc.), Woolworth Bldg., New York, N. Y.

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41 - This company had a contract concerning monazite sand with the Brazilian Government. The De Freitas properties are now being worked by the French Societe Miniere.
42 - Agent for the Société Minière, a French company in which Rezende & Co. is said to be interested.

The Roessler & Hasslacher Chemical Co., 10 East 40th St., New York, N. Y.
Rogers Brown & Crocker Bros. (Inc.), 21 East 40th St., New York, N. Y.
Wm. H. Taggart, 17 South Despiaines St., Chicago, Ill. (Buyer of silicate.)
Titanium Alloy Manufacturing Co., 94 Fulton St., New York, N. Y.; 6007 Euclid Ave., Cleveland, Ohio.
Varlacoid Chemical Co., 15 Moore St., New York, N. Y.
Vitro Co., 928 Fulton Bldg., Pittsburgh, Pa.

Foreign Countries

Algernon Lewin Curtis, P. O. 61, Westmoor Laboratory, Chatteris, England.
Deutsche Gasglühlicht Gesellschaft (Auergesellschaft), Berlin, Germany.
Etablissements G. Devineau, 26 Rue Lafayette, Paris, France.
Messrs. Augusto de Freitas (Ltd.), Hamburg, Germany.
David Gething, Landore Copper Works, Landore, S. O. Glamorgan, S. E. Wales.
L. le Personne & Co., 99 Cannon St., London, E. C. 4, England.
James H. Mason Smelting Co., 803 St. Clarens Ave., Totonto-2, Canada.

PATENTS RECENTLY REPORTED (1929 AND 1930)

BECKET, FREDERICK M. (To Electro Metallurgical Co.). Zirconium-treated Iron-Chromium Alloy. U. S. Patent 1,689,276, Oct. 30, 1928; Chem. Abs., vol. 23, No. 1, Jan. 10, 1929, p. 38.

Wrought articles such as tubing are formed of a ferrous alloy containing 0.008 per cent of zirconium, or a little more, and 10 to 60 per cent of cerium. The alloy has working properties superior to those of an alloy free from zirconium but otherwise of similar composition.

BECKET, FREDERICK M. Zirconium-treated Alloy. U. S. Patent 1,689,630; Foote-Prints, vol. 2, No. 1, 1929, p. 34.

A ferrous alloy containing 10 to 20 per cent of chromium and 0.003 per cent or more of zirconium is used for wrought articles such as tubing. The zirconium distinctly improves the working properties of the alloy.

DEUTSCHE GASGLÜHLICHT AUER-GES. Treating Ores of Titanium, Zirconium, etc. British Patent 291,004, May 23, 1927; Chem. Abs., vol. 23, No. 5, Mar. 10, 1929, p. 1099.

Ores of titanium and zirconium and other rare earths are sulphatized with H_2SO_4, and the resulting sulphates are roasted to form oxides. SO_2 and SO_3 evolved are passed over finely divided ores in the presence of O or air to form additional sulphates, and the partially sulphatized ore is then treated with H_2SO_4. The process may be carried out in two rotary furnaces.

DEUTSCHE GASGLÜHLICHT AUER-GES. M. B. H. Preparation of Pure Zirconium Sulphate from Zirconium Ores Decomposed by Sulphuric Acid. German Patent 434,987, Sept. 26, 1923; Chem. and Ind., vol. 49, No. 27, London, July 4, 1930, p. 612.

The concentrated, slightly acid solution of zirconium sulphate obtained by digesting the ore with sulphuric acid is treated with sulphuric or hydrochloric acid to precipitate zirconium sulphate. This is redissolved in dilute acid, the solution neutralized and boiled to precipitate a basic sulphate, and this salt is redissolved and reprecipitated by either of the above-mentioned methods to eliminate the last traces of iron and titanium.

DEUTSCHE GOLD-UND-SILBER-SCHEIDEANSTALT (To Roessler, 7, Weissfrauenstrasse, Frankfort-on-Main, Germany, and L. Weiss, 20, Mittelweg, Frankfort-on-Main, Germany). Tin, Titanium and Zirconium Oxides. British Patent 327,142, Dec. 27, 1928; Chem. Age, vol. 22, No. 570, London, May 31, 1930, p. 519.

Pigments and turbidity agents for vitreous enamels are obtained by treating soluble compounds of tin, titanium, and zirconium, with or without limited quantities of solvents, with substances capable of converting them into oxides, such as alkaline lyes, steam, or ammonia. The oxides or hydroxides obtained are converted into a voluminous form by heating to 500-800° C. in the case of pigments, or 900-1500° in the case of turbidity agents. The reaction may take place in the vapor stage, in which case the subsequent heating may be omitted. Reference has been directed by the comptroller to Specification No. 28,565/1908, 203,352, 206,284, and 296,730.

KARL, A. Zirconium Oxide. British Patent 314,526, Aug. 21, 1929; Jour. Am. Ceram. Soc., vol. 12, No. 11, Nov., 1929, p. 848.

Zirconium oxide is obtained from an alkali zirconate by bringing the latter into solution as sulphate and then precipitating the hydroxide, as by hydrolysis in neutral solution. Simultaneous precipitation of associated impurities such as iron, titanium, or aluminum is prevented by first reducing the solution with a powerful reducing agent such as nascent hydrogen, sulphurous acid, or sodium hyposulphite. It is stated that the process is applicable also to the production of metal oxides similar to zirconium oxide.

KIERNAN, WM. PHILIP. Preparation of Zirconium. U. S. Patent 1,760,413, May 27, 1930; Jour. Am. Ceram. Soc., Abs. Bull., vol. 9, No. 8, Aug., 1930, p. 652.

The process of producing formed articles of zirconium, which comprises producing a relatively pure coarse-metal powder of the metal, compacting the powder, heating the compacted article to a low temperature in a high vacuo to substantially degasify the compacted metal powder, sintering the degasified article at a high temperature approximately, but below the fusion point thereof in the same high vacuo, and thereafter subjecting the sintered article to mechanical deformation at temperature approximating 800° C.

I. C. 6456

June, 1931

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

ZIRCONIUM

PART II. DOMESTIC AND FOREIGN DEPOSITS

BY

E. P. Youngman

I. C. 6456
June, 1931.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE--BUREAU OF MINES

ZIRCONIUM[1]

PART II. DOMESTIC AND FOREIGN DEPOSITS

By E. P. Youngman[2]

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1 - The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6456."

2 - Rare metals and nonmetals division.

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FOREWORD

Zirconium, formerly distinctly a rare element, is rapidly becoming of rather extensive commercial importance. Economic features of the zirconium industry, together with general information regarding the metal and its compounds, are discussed in a separate paper,[3] to which this paper is a supplement.

In addition to giving a brief discussion of the four or five deposits that so far have proved to be commercial sources of the element, this circular summarizes available data (including some hitherto unpublished material from the files of the Bureau of Mines) upon other occurrences, including many of scientific interest only, of the two commercial minerals, zircon and baddeleyite, and also data upOn some of the minor minerals, such as eudyalite and cyrtolite, which have attracted additional interest in recent years because they are frequently richer in hafnium than the more common zirconium minerals.

NORTH AMERICA

United States and Territories

Alaska

Zircon has been found in black sand from Cape Nome, Eagle River, and Yukon Territory, Alaska.[4]

Arkansas

Lee[5] is authority for the statement that eudyalite (variant eucolite) has been reported in the United States in Arkansas alone. In 1861 Shepard[6] found at Magnet Cove, Ark., imperfect rounded crystals of eudyalite, imbedded in feldspar and associated with aegirine, all three minerals belonging to the extensive elaeolitic rock of that region. At first he thought the newly discovered mineral to be corundum, as its color was a rich crimson, varying to peach-blossom red. However, after testing its hardness (somewhat under 6), and after observing that it quickly gelatinized in chlorhydric acid, he decided that it was eudyalite.

3/ Youngman, E. P., Zirconium, Part I: Inf. Cir. 6455, Bureau of Mines,

4/ Pratt, Joseph Hyde, Zircon, Monazite, and Other Minerals Used in the Production of Chemical Compounds Employed in the Manufacture of Lighting Apparatus: North Carolina Geol. and Econ. Survey, Bull. 25, 1916, p. 18.

5/ Lee, O. Ivan, Mineralogy of Hafnium: Chem. Rev., vol. 5, No. 19, Am. Chem. Soc., 1928, pp. 29-35.

6/ Shepard, Charles Upham, Mineralogical Notices: Am. Jour. Sci. and Arts, ser. 2, vol. 37, May, 1864, p. 407.

In 1890 Williams[7] published an article upon the eudyalite and eucolite of Magnet Cove, confirming the conclusions of Shepard and others, and describing the crystals in detail.

The exact locality in which the minerals occur Williams described as follows:

> Both eudyalite and eucolite occur in two places in Magnet Cove: one, 100 meters northeast of the point where the Hot Springs turnpike crosses Cove Creek, and the other on "the branch" about 150 meters east of where it empties into Cove Creek. Indications of eudyalite have been observed in the elaeolite syenite of Saline County, Ark., on the property of Sol. Nethercutt, about 7 miles (11 kilometers) northeast of Benton (N. W. of S. E. of Sec. 16, 2S. 14W.). At this point the mineral was so badly weathered that an exact determination was impossible.
>
> Besides the aegyrite, elaeolite, and orthoclase of the rock in which the eudyalite occurs, there are found as accessory minerals beautiful little idiomorphic titanites and apatites. As decomposition products there appear ozarkite (thomsonite) and manganopectolite, which may be due in part at least to the weathering of eudyalite as well as of the other constituents of the rock.

Arizona

Pratt[8] reported zircon in Maricopa and Yavapai Counties. Day and Richards[9] reported the finding of zircon in the sands at Wickenburg, Maricopa County; at Arizona City, Skull Valley, and Prescott, Yavapai County; in the Santa Rita Mountains and at Greaterville, Pima County; and at Tucson, Pinal County.

California

Pratt[10] reports zircon in the following counties of California: Butte, Calaveras, Del Norte, Eldorado, Humboldt, Los Angeles, Nevada, Placer, Riverside, Sacramento, San Luis Obispo, San Mateo, Santa Barbara, Santa Cruz, Shasta, Siskiyou, Trinity, and Yuba. The counties of Plumas, San Barnardino, and Tuolumne are added by Day and Richards[11] in an article

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7/ Williams, J. Francis, Eudyalite and Eucolite, from Magnet Cove, Arkansas: Am. Jour. Sci., 3d ser., vol. 40, Nos. 235-240, July to December, 1890, pp. 457-462.

8/ Pratt, Joseph Hyde, Work cited, p. 18.

9/ Day, David T., and Richards, R. H., Useful Minerals in the Black Sands of the Pacific Slope: Min.Res. of the United States, 1905, U. S. Geol. Survey, 1906, pp. 1180-1181.

10/ Pratt, Joseph Hyde, Work cited, p. 18.

11/ Day, David T., and Richards, R. H., Work cited, pp. 1175-1258

upon the black sands of the Pacific Coast. This article, which shows that zircon stands fifth in point of frequency (quartz being omitted in the estimate), in the sands of the States investigated, gives not only the counties in California (and other States) in which zircon occurs but the number of pounds of zircon to a ton of sand in each locality.12/

Logan13/ says that zircon exists in small quantities as a primary constituent of the granodiorite that forms so large a part of the country rock east of the Sacramento Valley. In places where the sand is an accumulation from the erosion of the granodiorite, zircon is part of the heavy concentrates. No real work has been done so far to determine the practicability of recovering it.

Turner14/ discusses specimens of zircon crystals from heavy sands from gravel washings sent to him from a surface placer in Eagle Gulch, near Edmanton, about 4.5 kilometers southwest of Meadow Valley, Plumas County. Prof. S. L. Penfield, of the Sheffield Scientific School, who examined some of these crystals, reported as follows:

> The zircon crystals resemble ones that I have seen associated with gold from North Carolina. The forms are not unusual, 100, 110, 111, 311, and possibly 101. The hexagonal plates, referring to the crystals obtained by fusion with sodium carbonate (Na_2CO_3), resemble those I have met in testing for zirconia. I do not consider the reaction a very delicate one. The black grains may be chromite or some spinel mineral that would resist treatment with acids.

Colorado

Pratt15/ reported the existence of zircon in the following counties of Colorado: Chaffee, Costilla, Eagle, Huerfano, and Routt. Day and Richards reported zircon in the sands examined in these counties and in Gilpin, Jefferson, and Summit Counties as well. For details as to localities, see article by Day and Richards.16/ Lee reported cyrtolite in Colorado, though he stated that it is very sparing in occurrence.17/

Free made an examination of pitchblende from Colorado because, as he said, it is the only important uranium mineral stated to contain zirconium, and because among uraninites of the United States of America it is

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12/ Day, David T., and Richards, R. H., Work cited, pp. 1182-1190.
13/ Logan, C. A., Sacramento Field Division; Placer County; Zirconium: Dist. Rept. Min. Eng., Rept. 23 of the State Mineralogist, vol. 23, No. 3, California State Min. Bur., July, 1927, p. 286.
14/ Turner, H. W., Notes on Rocks and Minerals from California: Am. Jour. Sci., 4th ser., vol. 5, No. 30, June, 1898, p. 426.
15/ Pratt, Joseph Hyde, Work cited, p. 18.
16/ Day, David T., and Richards, R. H., Work cited, pp. 1190-1194.
17/ Lee, O. Ivan, Work cited, pp. 24-25.

unique in being analogous to the Joachimsthal pitchblende. His report is, in substance, as follows:[18]

> The lead-uranium ratio was very small and indicates a maximum age of 2,000,000 years. This establishes the recent formation of the mineral in contradiction to the other United States of America uraninites examined by Hillebrand and Boltwood. The identity of the constituent thought by Hillebrand to be zirconium has been confirmed, and no thorium or rare earths could be detected in the mineral. Analysis of the mineral gave 78.9 per cent U_3O_8, 5.25 per cent ZrO_2, with SiO_2, pyrites, and moisture as important constituents, and traces of lead, titanium, and vanadium. A comparison of the density of the zirconia separated from large quantities of the mineral with specimens of zirconia separated from a blue Siamese zircon, reputed to be rich in hafnium, and with a commercial zirconia preparation, showed that all these specimens were of similar low density and therefore of similar low-hafnium content. The spectra of the preparations were also similar and gave no evidence of any difference in hafnium content.

The zircon from the Pike's Peak district, Colo., has been described by Cross and Hillebrand.[19] In a prospect tunnel near the Pike's Peak toll road, almost directly west from Cheyenne Mountain, unusually beautiful transparent crystals were found in moderate abundance in a small quartz vein.

Connecticut

Cyrtolite occurs, although very sparingly, in Connecticut, the Branchville (Conn.) cyrtolite being mentioned by Lee[20] as one of the richest in hafnia in the United States.

Florida

The plant of Buckman & Pritchard (Inc.), at Mineral City, Fla., which had a recorded production of zircon from 1922 to 1928, ceased operation in 1929. The plant at Mineral City was first built for the recovery of ilmenite and was later adapted to the recovery of zircon and rutile also.

The composition of the concentrates of the Florida sands is as follows:[21]

18/ Free, O., Zirconium in Colorado Pitchblende: Phil. Mag., vol. 1, May, 1926, pp. 950-960. (Sci. Abs., vol. 29, Nov. 25, 1926, p. 820.)
19/ Cross, Whitman, and Hillebrand, W.F., Communications from the U. S. Geological Survey, Rocky Mountain Division, II; Notes on Some Interesting Minerals Occurring near Pike's Peak, Colo.; Zircon: Am.Jour. Sci., 3d ser., vol. 24, No. 142, Oct., 1882, pp. 284-286.
20/ Lee, O. Ivan, Work cited, pp. 24-25.
21/ Martens, James H. C., Beach Deposits of Ilmenite, Zircon and Rutile in Florida: 19th Ann. Rept. Florida State Geol. Survey, 1926-27, 1928, p. 137.

Minerals	Per cent
Ilmenite	55
Zircon...............	20
Rutile...............	6
Monazite.............	2
Greenspar............	14
Quartz...............	3

With respect to the occurrence of zircon in the beach sands at Mineral City, Fla., Martens says:22/

> Zircon is almost universally present in quartz sand, but the size of the grains and the proportion of them are so small that it can rarely be seen without using a microscope or without concentrating the mineral in some way, or both. However, on some of the beaches where natural concentration of the heavy minerals is going on, light-colored streaks of zircon may be seen among the darker rutile and ilmenite. The zircon can then be distinguished from quartz by its brilliant luster and smooth crystal faces. The zircon separated from the sand at Mineral City is in grains usually about 0.1 millimeter in diameter, but varying considerably in size and shape. Each of the grains is a single crystal; many of them show the crystal faces to a high degree of perfection, while others appear worn or broken. There are a few prismatic crystals as much as four or five times as wide, but crystals only slightly elongated are far more abundant. Practically all of the zircon on the Atlantic coastal beach from Mineral City to St. Simons Island (Georgia) is of the colorless variety, rarely showing zonal structures. Pale-purple or lavender grains of zircon are present but are very rare.

The following description of the beach deposit in general is also by Martens:23/

> From the mouth of the St. Johns River to St. Augustine Inlet, a distance of 35 miles, the ocean beach . . . is nearly straight and is rather uniform in its topographic features . . . The width of the beach for most of this distance is about 500 feet at low tide. At the back of the beach is a line of dunes usually a few hundred feet wide and varying in height up to 30 feet or a little more. A large part of the way along the beach there is a distinct wave-cut cliff or notch in the dune front, which has been produced during storms, since it is not reached at high tide in calm weather. For many miles inland none of the land is as high as the highest dunes, and from Mineral City southward to St. Augustine Inlet, there are marshes immediately in back of the dunes.

22/ Martens, James H. C., Work cited, p. 133.
23/ Martens, James H. C., Work cited, pp. 127-129.

The heavy concentrate or ore always occurs in beds, which may best be described as "strips" since the bodies are much wider than they are thick and much longer than they are wide, the direction of greatest extent being parallel to the shore. The principal beds occur on the back part of the beach at the foot of the dunes and extending out a short distance in front of them. . . . The ore is in the form of a fine sand, even finer than the ordinary beach sand, and is easily recognized by its dark color and heavy weight. A cut through the back part of the beach will usually reveal a succession of thin light and dark beds, the dark ones varying in thickness from a small fraction of an inch up to about 6 inches, while the light ones are generally thicker. . . . There is frequently a fairly thick layer of black sand almost immediately below the surface, while deeper down there is a succession of thinner layers of light and dark.

During the early period of working, the richest streak of black at the foot of the dunes contained 60 per cent of "mineral" (heavy minerals recovered in the wet mill) for a thickness of 2 to 2½ feet and a width of 25 to 35 feet, but the average of the sand as worked has recently been around 20 per cent "mineral" and the dimensions as worked somewhat smaller. Samples can be obtained that are practically free from quartz and therefore may be said to contain 100 per cent "mineral," but such material is in too thin layers to select it in mining; so a certain amount of the quartz sand has to be mined also. All of the dunes and beach sand contain some of the ilmenite and associated minerals; so that there is not always a sharp distinction between material that is workable and that which is not. Besides the beds on the back part of the beach, some of the adjoining part of the dunes selected as appearing richest in dark minerals has also been worked.

The workings have extended 3 miles north of the mill at Mineral City and 8 miles south. To the north the deposits continue somewhat farther, but the beach is developed as a pleasure resort to such an extent as to prevent mining operations. Farther to the south the heavy concentrate probably is of workable grade and depth for several miles and is likely to be mined when a road is built so as to make it more readily accessible.

Both the heavy concentrates and the associated beach and dune sand are entirely loose and unconsolidated and of very fine texture. The sand of the lake beaches is generally coarser than that of the ocean beach. The same heavy minerals are present in the ordinary beach and dune sand as in the heavy concentrate, but in much smaller amount. There are all gradations from the dark heavy concentrates with only a few per cent of quartz to sand that is practically all quartz with only scattered grains of the heavy minerals. . . .

Microscopic examination is the only practicable method of determining the mineral composition since there is too complicated a mixture for chemical analysis to be of much help, and anything approaching complete separation of the minerals by mechanical means is excessively difficult.

Other occurrences of zircon in Florida, which are of no commercial value as yet, as reported by Martens[24] are as follows:

1. Amelia Island.--Specimens from heavy concentrates from the beach and dune sands of Amelia Island, at the extreme northeastern corner of the State, show by analysis 11.3 and 24.1 per cent of zircon.

2. Eau Gallie.--One and one-half miles north of the main part of Eau Gallie, on the west side of Indian River, the beach has heavy mineral concentrates, as do also the beaches at several other places on Indian River. The percentages of zircon in specimens tested are 14.7 and 15.

3. Gulf Beach at Venice.--For a distance of 2 miles or more south of Casey's Pass, on the gulf beach near Venice, the heavy minerals are abundant. The zircon, being the heaviest of the minerals present in any large amount, is left highest on the beach by the waves. Some of the rich zircon sands (scraped from a layer a fraction of an inch in thickness) yielded 68.6 and 69.4 per cent of zircon. The more common deposits of concentrates at Venice yielded 6.6 and 11.2 per cent of zircon.

4. Cape San Blas.--Heavy mineral concentrates extend for some miles along the upper part of the gulf beach on the peninsula between Cape San Blas and Point St. Joseph. The deposit is not wide and thick enough to be commercially important under present conditions. Analyses show 5 and 15.7 per cent of zircon in the sands.

5. Crooked Island.--On the mainland side of Crooked Island (Bay County), along St. Andrews Sound, are deposits of heavy concentrates from about low tide to above high tide; and on the outer beach of the island, almost opposite the end of the road from Auburn to St. Andrews Sound, are thin layers of heavy concentrates, which alternate with quartz sand. Examination of the island did not disclose deposits large enough to be workable; however, but a small part of the island was included in the investigation. Specimens analyzed showed 17.9 and 20.5 per cent of zircon

6. Inlet Beach.--In Walton and Bay Counties, on the gulf beach, for a distance of 2 miles to the east and 5 miles to the west of Phillips Inlet, the darker minerals are found in the sands, although

24/ Martens, James H. C., Work cited, pp. 141-147.

the light sands predominate. On the back part of the beach layers of heavy dark concentrates alternate with white quartz sand. The composition of the sand is similar to that from Crooked Island and Cape San Blas. However, there is less zircon here than in the other two localities.

7. Santa Rosa Island.--Heavy sands, the composition of which is similar to that of concentrates on Crooked Island, occur in thin narrow strips along the shore on the north side of Santa Rosa Island, opposite Camp Walton.

8. Lake Beaches.--Deposits of heavy mineral concentrates, not extensive enough to justify exploitation, exist on many lake beaches, such as Barnes' Beach, on the north side of Lake Weir, Marion County (2.1 and 3.5 per cent of zircon); east side of Lake Geneva, Clay County (40.2 and 57.6 per cent of zircon); and Kingsley Lake Beach, at Strickland's resort, Clay County (0.3 and 0.4 per cent of zircon).

Georgia

According to Teas,[25] on a number of islands along the Atlantic coast of Georgia (as well as of Florida) black sands have been found in which the content of rare minerals is believed to be sufficiently large to warrant their consideration as commercial deposits.

At the southern end of St. Simon Island (which lies northeast of Brunswick, and which is reached by boat from that city), near the wharf and in front of the lighthouse, occur the heaviest concentrates, extending almost half a mile. The dark minerals are on the upper part of the beach sands, between high and low tide marks. Back from the sea the black sand is covered by wind-shifted sands to a depth of 1 to 10 feet. An analysis of the sand from St. Simon showed 0.12 per cent of zirconium oxide.

On Sapelo Island, which is reached by boat from Darien, the black sand is most plentiful a short distance north of the lighthouse, which is at the south end of the island. It is found, however, over most of the island, especially along the beaches--the dunes not containing much of the darker sands. An analysis of a specimen from the richest deposit, north of the lighthouse, which showed 5 to 6 per cent of the dark minerals, indicated the presence of quartz, ilmenite, magnetite, monazite, and zircon, the relative abundance thereof being indicated by the order in which they are given. The percentages of zirconium oxide in two tested samples were 0.08 and 0.1.

In Charlton County, 3 miles west of St. George, sand similar in mineral content to that of the coastal islands was reported to be associated with deposits of sap brown along the Georgia and Florida Railway.

[25] Teas, L. P., Sand and Gravel Deposits of Georgia: Geol. Survey of Georgia Bull. 37, 1921, pp. 376-377.

Idaho

The report of Day and Richards upon the minerals in the black sands of the Pacific slope[26] lists the following counties of Idaho in which zircon occurs: Ada, Bannock, Bingham, Blaine, Boise, Canyon, Custer, Elmore, Fremont, Idaho, Lemhi, Lincoln, Nez Perce, Owyhee, Shoshone, and Washington. The individual sand deposits are too numerous to mention here.

Minute crystals of zircon are prevalent in the metamorphosed rocks of the Pend Oreille mining district, an area 15 by 20 miles; and in the nearby Coeur d'Alene district, also, zircon is very abundant in the sedimentary rocks that are directly correlated with the pre-Cambrian formations of the Pend Oreille district, as shown by Joseph L. Gillson, from whose report[27] upon zircon as a contact metamorphic mineral in the Pend Oreille district the following excerpts are taken:

> The sedimentary rocks of the district belong to the Belt series (Algonkian) and the Cambrian. Except for one calcareous member, the Belt rocks consist of a great thickness of thoroughly indurated sandstones and shales, and the Cambrian rocks consist of quartzite, shale, and limestone. A very large batholith extends from the west side of the district westward for many miles, and three stocks, each from two to three miles across, outcrop within the area. These igneous bodies are so large and are so near to each other even at the surface that a widespread effect of igneous metamorphism would be expected in the invaded sedimentary rocks. In fact no locality in the quadrangle was found where the sedimentary rocks did not show at least some microscopic evidence of igneous metamorphism.
>
> Detrital zircon is commonly present in the noncalcareous rocks; but little or none is present in the calcareous rocks. . . .
>
> The evidence from the calcareous rocks, more striking than that in the noncalcareous rocks, leaves no room for doubt that zircon is a contact-metamorphic mineral. The sequence of the mineralization under the contact metamorphic conditions in the sedimentary rocks of the Pend Oreille district indicates that the zircon belongs to a second or pneumatolytic stage in the metamorphism and formed contemporaneously with tourmaline, biotite, andalusite, cordierite, vesuvanite, garnet, diopside, apatite, etc. Unlike these minerals the zircon is of unrestricted occurrence and is found in both calcareous and noncalcareous rocks.

26/ Day, David T., and Richards, R. H., Work cited, pp. 1194-1200.
27/ Gillson, Joseph L., Zircon, a Contact Metamorphic Mineral in the Pend Oreille District, Idaho: Am. Mineral., Jour. Mineral. Soc. Am., vol. 10, No. 8, Aug., 1925, pp. 187-194.

Indiana

Zircon was found (in the proportion of 66 pounds to a ton of sand) in sands from Michigan City, Laporte County, Ind.[28]

Maine

Zircon is known to occur in pegmatite at Auburn, Me. In the town of Norway, near Cobble Hill, perfect crystals of zircon, with chrysoberyl and zinc spinel, occur in the pegmatite. Zircon crystals one-sixteenth to one-eighth inch long lie upon slickensided surfaces, of which Bastin says, "They were probably formed during the shearing process." Zircons, associated mostly with triphyllite and rarely exceeding one-eighth inch in diameter, have been found in the pegmatite of Mount Mica, Paris, Me.[29]

Maryland

Zircon was found in sands examined in Maryland: from a trace at Glyndon to 2 pounds per ton of sand at Harrisonville (Baltimore County); and 19 pounds per ton at Ocean City, in the beach sand (Worcester County).[30]

Massachusetts

The cyrtolite at Rockport, Mass., although very rare, is unusually rich in hafnia, containing as much as 17 per cent thereof.[31]

Mississippi

A trace of zircon was found in sands examined at Magnolia, Pike County, Miss.[32]

Montana

Zircon has been reported as occurring at Wisdom, Beaverhead County; Miles City, Custer County; and in Powell County, Mont.[33]

Austin F. Rogers, while examining specimens of corundum from an unknown locality in Montana (obtained from Ward's Natural Science Establishment), discovered a black submetallic mineral, which he proved to be baddeleyite. Baddeleyite has been identified by Rogers in specimens from the property of the Bozeman Corundum Co., 14 miles southwest of Bozeman. The baddeleyite of Montana is an accessory constituent of a gneissoid

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28/ Day, David T., and Richards, R. H., Work cited, p. 1200.
29/ Mining and Engineering World, ---: Vol. 37, 1912, p. 1018.
30/ Day, David T., and Richards, R. H., p. 1202.
31/ Lee, O. Ivan, Work cited, pp. 24-25
32/ Day, David T., and Richards, R. H., Work cited, p. 1202.
33/ Day, David T., and Richards, R. H., Work cited, p. 1202.

corundum-syenite containing microcline-microperthite, biotite, and corundum, with subordinate amounts of muscovite, sillimanite, and zircon.34/

New Hampshire

According to Pirsson and Washington,35/ microscopic study of the foyaite type of nephelite syenite reveals zircon in thin sections at Red Hill, Moultonboro, N. H. The zircon, not very common, is in rough crystals or grains that have a general though sparse distribution. The crystals vary from 0.2 to 0.4 millimeter in diameter. The zircon is included with the accessory minerals in the umptekite variety of alkalic syenite.

The first occurrence of wohlerite in an American locality (a well-known mineral characteristic of the nephelite-syenite pegmatite dikes of South Norway) was reported at Red Hill, where it is found everywhere distributed through the rock mass.

New Jersey

A report on the zircon of New Jersey by the United States Geological Survey36/ is quoted at length.

> Several localities in northern New Jersey, especially in Sussex County, have been reported to contain zircon-bearing ores and rocks, and an investigation of the deposits was undertaken to test their availability as a source of zircon. It has been concluded that if the demand for zircon should become imperative and if the mineral must be had at any cost, then several hundred tons could be obtained from northern New Jersey, but the locality does not offer any inducement as a commercial field. . . . . The most promising pegmatite is that of the old Woods mine, about half a mile southeast of Stockholm, N. J. At least four shafts, now filled in, were made in a pegmatite dike about 10 feet wide and exposed for more than 100 feet. The pegmatite rock is composed of quartz, feldspar, magnetite, and zircon. There is too little magnetite in the rock to be worked as an iron ore. The shafts are said to have been dug long ago--maybe over 100 years. No ore is known ever to have been shipped away. The rock is richer in zircon than any other rock seen in this region. Some selected parts of the dike may carry as much as 5 per cent of zircon, but an average sample of 20 specimens of selected zirconiferous rock yielded only 1.53 per cent of zircon. The rock in the shaft at one end is much richer in zircon than the rock at the other end of the line of shafts. An average sample of 10 specimens, collected across the exposed dike at 1-foot intervals, con-

34/ Rogers, Austin F., Baddeleyite from Montana: Am. Jour. Sci., 4th ser., vol. 33, No. 193, Jan., 1912, pp. 54-56

35/ Pirsson, L. V., and Washington, H. S., Contributions to the Geology of New Hampshire, No. 3; On Red Hill, Moultonboro: Am. Jour. Sci., 4th ser., vol. 23, No. 136, April, 1907, pp. 261, 266, 267, and 270.

36/ Schaller, Waldemar, T., Thorium, Zirconium, and Rare-Earth Minerals: Min. Res. of the United States, 1919, pt. 2, U. S. Geol. Survey, 1922, pp. 21-22.

tained 0.92 per cent of zircon. The pegmatite rock is hard and unaltered and would have to be crushed before any zircon could be obtained. The total quantity of zircon obtainable from this locality would not amount to more than a few hundred tons, and the expense would prohibit commercial exploitation.

Nevada

Traces of zircon were found in the sands of Dixie Creek and of Mascot, Elko County, and of Berlin, Nye County. Zircon was found at Carson City, Ormsby County, also.[37]

New Mexico

Day and Richards[38] reported the occurrence of zircon in the sands at Pinos Altos, Grant County; at Brice, Otero County; at Bernardillo, Sandoval County; and at Los Cerrillos and from Tuer to Arroyo, Santa Fe County.

New York

Morris says[39] that zircon occurs in New York, at Lyon Mountain, Clinton County; at a few places near Crown Point and abundantly in pegmatites at Old Bed Mines, Mineville, Essex County; and at numerous places in Orange County, on the south, and St. Lawrence, on the north. Day and Richards[40] report small amounts of zircon found in the sands examined at Lowville and Rochester, Lewis County.

A deposit of cyrtolite, radioactive, in Westchester County, which Lee says has been twice authenticated as high in hafnia,[41] has been fully described in a circular upon hafnium, soon to be published by the Bureau of Mines.

North Carolina

North Carolina (the only State with the exception of Florida that has had a recorded production of zirconium ore in the United States) was the first producer of the ore. The silicate zircon was discovered in Henderson County in 1869. Production continued intermittently from 1869 until 1911. W. E. Hidden, according to Lee,[42] mined as much as 26 tons as early as 1883. The first shipment recorded, however, was in 1888,

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37/ Day, David T., and Richards, R. H., Work cited, p. 1204.
38/ Day, David T., and Richards, R. H., Work cited, p. 1204.
39/ Morris, H. C., Zirconium: Political and Commercial Control of the Mineral Resources of the World, No. 7, Bureau of Mines, Nov. 15, 1918, pp. 9-13.
40/ Day, David T., and Richards, R. H., Work cited, p. 1204.
41/ Lee, O. Ivan, Work cited, pp. 24-25.
42/ Lee, O. Ivan, Work cited, pp. 24-25.

when the ore was marketed to be used in the manufacture of mantles for incandescent lights. As thoria soon took the place of zirconia for this purpose, shipments of zircon ceased for a few years, beginning again in 1902, upon the introduction of the Nernst lamp. Although the North Carolina Geological Survey in 1916 reported[43] that shipments begun in 1902 had continued until 1916, no one year having a very large production, no output figures were reported to the United States Geological Survey.

The following description of the Henderson County deposit is by Schaller.[44]

> * * * Near Tuxedo (formerly called Zirconia) a pegmatite dike, about 100 feet wide and striking N. 50° E., cuts through the pre-Cambrian gneisses of the region and has been traced for a mile and a half. The upper part of the pegmatite is kaolinized and disintegrated to a depth of 40 feet or more. Zircon crystals are present in abundance in certain parts of the pegmatite but are not uniformly distributed throughout the dike. They are gray in color and show both the prism and pyramid about equally developed. They average in size from about an eighth to a quarter of an inch. They can be readily washed from the decomposed pegmatite or from the unaltered crushed rock.
>
> Two places have been worked on this pegmatite dike--the Freeman mine, near the southwest end of the dike, and the Jones mine, near the northeast end. * * *

Genth[45] described other occurrences of zircon in North Carolina as follows:

> Abundant with the gold sands of Burke, McDowell, Polk, Rutherford, Caldwell, Mecklenburg, Nash, Warren, and other counties, in very minute yellowish-brown and brownish-white, sometimes amethystine and pink crystals with many planes. . . . Found also by Dr. Hunter at Well's farm, Gaston County. It is rarely found at Ray's mine, Hurricane Mountain, Yancey County, and the Flat Rock mine, Mitchell County. It has been observed in dark red-brown crystals in the magnetite beds of the Unaka Mountains; an irregular large crystal of about 2 inches in length and a pale brownish-gray color has been found by J. A. D. Stephenson near Statesville, Iredell County; and by the same, small crystals embedded in allanite, near Bethany church.
>
> Peculiar dark brown crystals from 1-3 millimeters in size are found at Low's and Tibbet's mine, in Macon County, which may be zircon. They need fuller investigation. Hidden reports, from the

43/ Pratt, Joseph Hyde, Work cited, p. 16.
44/ Schaller, Waldemar T., Work cited, p. 19.
45/ Genth, Frederick Augustus, The Minerals of North Carolina; U. S. Geol. Survey Bull. 74, U. S. Geol. Survey, 1891, p. 49.

gold sands of Brindletown, good crystals of the variety malacone. The latter are jet black, with occasionally a grayish crust, and are larger than, and of different form from, the zircons directly associated with them. Specific gravity 4.087. The same authority reports the variety cyrtolite from several places, namely:

> Masses and distinct crystals having curved faces and gray-brown color have been met with at the Wiseman mica mine in Mitchell County, associated with autunite, fergusonite, and samarskite. Also at Mill's mine, near Brindletown, and at the xenotime and polycrase locality on the Davis land, near Green River, in Henderson County.

With the monazite at Mars Hill, Madison County, zircon crystals of considerable size are sometimes found. One such crystal, specific gravity 4.507, was analyzed by me as follows:

Loss on ignition	1.20
Silica	31.83
Zirconia	63.42
Ferric oxide	3.23
	99.68

Oklahoma

Mineral Resources of the United States for 1907[46] announced that a new deposit of zircon had been reported by Frank Rush, a forest supervisor of the Wichita National Forest. This deposit (which was later prospected by Hackney & Sons, La Harpe, Kans.) has been described as follows:[47]

> A pegmatite dike with many scattered zircon crystals occurs near the south edge of the Wichita National Forest, Wichita Mountains, about 7 miles northwest of Cash, Okla. The zircon crystals reach a maximum size of nearly an inch, though most of them are much smaller. They are simple pyramids with the prism faces nearly absent. Most of the crystals are deep reddish-brown, but a few are yellowish to nearly colorless.

An investigation of the deposit seemed to indicate that only a very small quantity of zircon could be obtained, as the zircon-rich portion of the pegmatite was of slight extent.

46/ Sterrett, Douglas B., Monazite and Zircon: Min. Res. of the United States, 1907, pt. 2, U. S. Geol. Survey, 1908, pp. 792-793.
47/ Schaller, Waldemar T., Work cited, pp. 22-23

Day and Richards[48] reported traces of zircon in the sands at Putnam, Dewey County, Okla.

Oregon

Zircon occurs in the following counties of Oregon, according to a report of Pratt:[49] Baker, Clatsop, Coos, Lincoln, Curry, Douglas, Grant, Jackson, Josephine, Lincoln, Linn, Malheur, Multhomah, Polk, Umatilla, Wasco, and Washington. For the definite localities (far too numerous to mention here) in the counties just listed and in other counties, the reader is referred to the article by Day and Richards,[50] which has been quoted frequently in this paper.

Pacific Slope

That zircon stands fifth in the order of frequency (among 37 minerals in addition to quartz) in the black sands of the Pacific slope is claimed in a report[51] of the United States Geological Survey, which has been frequently referred to in these pages. A table covering 42 pages gives detailed information, by counties and smaller political divisions, concerning the occurrence of zircon (as well as of a dozen other minerals) on the Pacific slope. Magnetite, gold, ilmenite, garnet, hematite, chromite, platinum, iridosmium, mercury, amalgam, olivine, and iron silicates, pyrite, monazite, copper, cinnabar, cassiterite, and corundum are the other minerals most frequently found in the sands.

The action of a Wetherill separator on the zircon crystals is given on p. 1234, of Day and Richards' report.

Pennsylvania

Zircon is found in the following localities in Pennsylvania:

Berks County: Bernharts, Pricetown, and Trexler mica mine (Alsace).

Bucks County: Finney's quarry, Neshaminy Falls, Siles, and Vanartsdalen's quarry.

Chester County: Chester Springs, Copesville, Pughtown, Springton, Octoraro Creek, West Chester (Bath Springs), and Willowdale.

Delaware County: Avondale (cyrtolite), Blue Hill, Boothwyn (cyrtolite), Brandywine Summit, Painter's farm on Dismal Run, and Morgan Station (cyrtolite).

Lehigh County: Macungie.

Montgomery County: Lafayette, Willow Grove.

Northhampton County: Chestnut Hill (gray or pinkish crystals 5 centimeters long).

48/ Day, David T., and Richards, R. H., Work cited, p. 1204.
49/ Pratt, Joseph Hyde, Work cited, p. 18.
50/ Day, David T., and Richards, R. H., Work cited, pp. 1206-1215.
51/ Day, David T., and Richards, R. H., Work cited, pp. 1175-1258.

Philadelphia: Broad Street and Olney Avenue (cyrtolite), Bridesburg (colorless crystals in Delaware River sands), Comley's quarry (Mount Airy, cyrtolite), Fairmount Park, and in the Schuylkill River sands and gravels.

The following analysis is of zircon from Pricetown, Burks County, found as chocolate-brown crystals in magnetite:

	Per cent
SiO_2	34.07
ZrO_2	63.50
Fe_2O_3	2.02
H_2O	.50
	100.09

The specific gravity of the specimen of which the analysis is given is 4.595.52/

South Carolina

Zircon occurs in South Carolina associated with magnetite sand or ore. Venable mentions an occurrence at Anderson, S. C., as being among the three localities in the United States where the mineral occurs in quantities sufficient for being mined.53/

South Dakota

Day and Richards54/ reported zircon in the sands at Tinton and in the Hurricane district, Lawrence County; at Sheridan, Pennington County; and in the Big Horn Mountains.

Tennessee

Zircon occurs in Tennessee, associated with magnetite ore or with sand.55/

Texas

Lee56/ lists Texas among the States found to be the richest in hafnium minerals, the altered zircon, cyrtolite, being found in abundance in Llano County. As early as 1889, Hidden and Mackintosh57/ described the cyrtolite of Texas as follows:

52/ Gordon, Samuel G., The Mineralogy of Pennsylvania; Descriptive Mineralogy of Pennsylvania: Special Pub. 1, Acad. Nat. Sci. of Philadelphia, 1922, p. 90.
53/ Venable, Francis P., Zirconium and Its Compounds: Am. Chem. Soc. Monograph Ser., Chemical Catalog Co. (Inc.), 1922, p. 18.
54/ Day, David T., and Richards, R. H., Work cited, pp. 1215-1216.
55/ Venable, Francis P., Work cited, p. 18.
56/ Lee, O Ivan, Work cited, pp. 24-25.
57/ Hidden, W. E., and Mackintosh, J. B., A Description of Several Yttria and Thoria Minerals from Llano County, Texas: Am. Jour. Sci., 3d ser., vol. 38, Nos. 223-228, 1889, pp. 485-486.

Cyrtolite has been found abundantly in both massive form and in good crystallizations. One hundred kilos have thus far been collected while mining the yttria minerals already herein described. This mineral here occurs in thick plates attached to the biotite and also constitution veins in the coarse pegmatite. It is often the matrix of the thoro-gummite and fergusonite. Specific gravity is 3.653. It occurs in tetragonal forms with all the planes rounded, and polysynthetic groupings of crystals are very common. Its color ranges from dull gray, through various shades of brown, to deep brown and almost black. Hardness about 5. . . .

Hidden,58/ a few years later, said:

Many hundred pounds of cyrtolite were found and in great variety of form and color. All kinds of it gave good radiographs after 24-hour exposures. Plates of it as large as one's hand, covered on one side with curved crystals, were not rare. It sometimes encrusted large quartz crystals to the depth of 1 inch, having radiate structure, and thus afforded a new feature for this mineral and one very uncharacteristic of zircon. . . .

Utah

Zircon was found in the sands examined at Hite, Garfield County, Utah; at Sand Springs, Iron County; at Morgan, Morgan County; at the junction of the Combwash and San Juan Rivers, San Juan County; in Green River, Gensen district, and near Gensen, Uinta County.59/

Virginia

Zircon has been discovered in both sedimentary and igneous rocks in Virginia, occurring in sandstone near Ashland and in pegmatites near Gouldin, Hanover County, and near Amelia Courthouse, Amelia County. The zirconiferous sandstone and the zircon-bearing pegmatites of Hanover County are less than a mile apart, the former being a "part of the western edge of the Coastal Plain, near and along the overlap of the sediments upon the older crystalline rocks of the Piedmont Plateau."60/ Under the heading "Genesis," the relation between the two different types of deposits in Hanover County is shown by Watson and Hess, as follows:61/

The zircon and ilmenite concentration evidently represents an old beach segregation along but within the western margin of the

58/ Hidden, William E., Some Results of Late Mineral Research in Llano County, Texas: Am. Jour. Sci., ser. 4, vol. 19, No. 114, June, 1905; pp. 432-433.
59/ Day, David T., and Richards, R. H., Work cited, p. 1216.
60/ Watson, Thomas L., and Hess, Frank L., Zirconiferous Sandstone Near Ashland, Virginia: U. S. Geol. Survey Bull. 530-P, 1912, p. 3.
61/ Watson, Thomas L., and Hess, Frank L., Work cited, pp. 7-8.

Miocene sediments of the Coastal Plain, of Calvert age, and is similar to the black-sand beaches of New Jersey, California, Oregon, and numerous other coasts and to the gold-bearing garnet (so-called "ruby") sands of the beaches at Nome, Alaska.

The zircon and other heavy minerals resistant to atmospheric agencies were derived by weathering processes from the crystalline rocks, chiefly granites and gneisses, of the Piedmont Plateau, which extend westward from the Coastal Plain contact . These formed the country rock of the shore, and the zircon and associated minerals derived from them by weathering were accumulated by waters near the mouth of a small stream or behind a sheltering point, while the quartz sand was largely worn and carried away by the currents of the sea.

Zircon is an almost constant minor accessory mineral in the crystalline rocks of this old shore and its extension westward, and in places it occurs in large masses. Near Gouldin post office, 10 to 15 miles southwest of the Ashland area, pieces of zircon 3 inches in diameter weathered out of pegmatite dikes have been noted on the surface. Massive zircon without crystal outline and measuring 6 by 4 inches has been observed in the pegmatites of Amelia County, Va. Similar dikes occur in the gneiss-granite complex of the Piedmont Plateau, forming the old shore line which extends entirely across Virginia from Maryland into North Carolina, roughly coinciding with the meridian of 78° 30'. It seems probable that similar zircon-rich rocks may occur at numerous points along this old shore line. Many zircon-bearing deposits may be covered by later sediments and some may have been removed by erosion, but it is probable that others, which may be richer or poorer, will be discovered along the contact of the granite and gneiss of the Piedmont Plateau with the overlying sediments of the Coastal Plain.

Zircon-bearing sandstone deposit.--The Ashland deposit has been described by Schaller, as follows:62/

A bed of zirconiferous sandstone is exposed about 3 miles west of Ashland, Va. The bed does not crop out as a continuous ledge, but is represented on the surface by isolated flat fragments or bowlders, only a few of which are as much as a foot long. The largest bowlder seen measures 26 by 15 by 10 inches; the average diameter of the bowlders is about 4 to 6 inches. These isolated fragments and bowlders are found in the clay, gravel, or sand soil for about a mile north and south and about 500 feet east and west. The vertical thickness of the zone containing these bowlders does not seem to be more than a few feet, although there is almost no evidence on this point. The zirconiferous bowlders seen on the surface would not weigh altogether more than several hundred tons.

62/ Schaller, Waldemar T., Work cited, pp. 20-21.

The zirconiferous bowlders occur at the "fall line," or junction of the coastal sediments and the igneous rocks of the Piedmont, which are less than a mile west. The hard brownish bowlders are held to represent a local cementation of a soft sandy bed which was found in the lower part of a well 14 feet deep near the home of Benjamin Wright, 3/8 of a mile southwest of the Shelton home, where these hardened surface bowlders occur in greatest numbers. The sand in Mr. Wright's well contained 13 per cent of zircon. The hardened bowlders found on the surface for a distance of nearly a mile showed a greatly varying content of zircon, the maximum being 30 per cent.

The compact sandstone contains much ilmenite and quartz and smaller quantities of rutile, staurolite, kyanite, feldspar, and other minerals. All of it is cemented by brown limonite. The density of the sandstone is a good indication of its zircon content; for the pieces very poor in zircon weigh perceptibly less than those rich in zircon. A collection of 24 samples of the brown bowlders from the northernmost exposure contained only 0.5 per cent of zircon; 7 samples from another place yielded 3 per cent of zircon; 19 samples from another place gave 12 per cent of zircon; a compact brown bowlder near Mr. Wright's house contained 25 per cent of zircon; and the average content of 32 pieces of fine-grained sandstone from the Shelton farm contained 25 per cent of zircon. On the other hand, 17 samples of coarse-grained sandstone from the Shelton farm, similar in appearance to the fine-grained material except in the size of its particles, averaged only 1 per cent of zircon.

Ten samples of the clay dirt collected from the well on the Shelton farm, at 2-foot vertical intervals, contained zircon from a trace to nearly 0.5 per cent. Only three of the samples yielded more than 0.1 per cent of zircon, and five of the samples had less than 0.03 per cent.

The occurrence was thoroughly tested in 1912 by G. L. English, who sunk a number of pits on the Shelton farm. In one of these pits a solid bed of the zirconiferous sandstone was found 7 feet below the surface; in the other pits only isolated fragments of the hardened sandstone were found. There seems to be very little evidence of the existence of a continuous bed of this sandstone, short lenses a few feet in length seeming to be the general feature. Most of these lenses have been broken up into isolated fragments and bowlders, and there is almost no evidence of a continuous bed of this particular rock. Moreover, the diverse character and zircon content of the bowlders, as found, show that only a part of the deposit contains enough zircon to be considered a possible source of that mineral.

The very evidently stratified character of the bowlders, many of which have a distinctly layered structure, indicates that the local cementation extended for horizontal distances of about 10 feet

or more. The problem of the presence or absence of a distinct well-defined bed a mile long can be solved only by a detailed study of the region, involving the sinking of numerous pits.

Zircon-bearing pegmatite deposits.--In the well-known pegmatites near Amelia, Amelia County, and near Gouldin, Hanover County, in the middle-eastern part of the Piedmont Plateau province, separated by a distance of approximately 40 miles, are large masses of zircon. According to Watson,63/ the features of special interest in connection with the pegmatites of the two localities are: (1) its occurrence in massive forms of unusual size, and (2) its association in the two places with an entirely different group of the rarer minerals, although the pegmatites of each area are of granitic composition. Zircon and apatite are the only rare minerals that have been found alike in the two areas, and so far as it has been determined the zircon is more abundant in the Hanover County deposit than in that of Amelia County.

The following descriptions (given in part) of the two areas are by Watson:64/

> Amelia County area.--The pegmatite bodies occurring near Amelia County have long been known for the variety of rare minerals found in them, many of which were of unusual size. The dikes have been worked from time to time for a long period of years as a source of commercial mica and feldspar, and to a less extent of minerals for the gem trade.
>
> The country rock is a thinly foliated, moderately dark-colored, fine-grained biotite gneiss or schist, containing more or less muscovite. Where measured, the foliation strikes N. 25° to 30° E. and dips 40° to 50° N. W. Diabase dikes of Mesozoic age intrude the rocks in places. The pegmatite bodies are dikelike in form and nearly vertical, with the direction of trend doubtful. They cut across the foliation of the schists, and the large ones will measure more than 50 ft. across. They are cut by joints, but there is no evidence of schistose structure developed from metamorphism.
>
> The pegmatites are of granitic (acidic) composition, containing feldspar, including the potash varieties, orthoclase and green microcline, and the soda variety, albite, with quartz and muscovite, and a large number of rarer minerals. The principal rock-forming minerals are not uniformly distributed through all parts of the pegmatites, but their distribution is very irregular, first one and then another of these minerals predominating in different parts. The albite, occurring in splendid crystallizations as reticulated platy forms of bluish white to white color and frequently transparent, is of a high degree of purity. . . .

63/ Watson, Thomas L., Zircon-Bearing Pegmatites in Virginia: Am. Inst. Min. Eng. Bull. 115, July, 1916, p. 1238.
64/ Watson, Thomas L., Work cited, pp. 1237-1243.

The texture of the pegmatites is granular consertal rather than graphic. Miarolitic cavities have been observed in some of the openings made in the pegmatites. One of these was of large size, the walls of which were lined with crystals of smoky quartz and pure white crystals of albite, some as transparent as glass.

The rarer minerals include representatives of five distinct chemical groups: (1) Halcids, including fluorite; (2) silicates, including garnet (spessartite), black tourmaline, beryl, helvite, allanite, and zircon; (3) niobates, including columbite; (4) tantalates, including microlite; and (5) phosphates, including apatite and monazite. With the exception of fluorite, tourmaline, and zircon, each of the minerals has been analyzed with the results shown below. Some of these minerals have been found only occasionally in the Amelia pegmatites and are very rare. Many of them attained unusual size, such as crystals of beryl 3 to 4 feet long and 18 inches thick, columbite in crystalline masses weighing 6 to 8 pounds, allanite crystals more than 15 inches long, microlite in masses up to 8 pounds in weight, and monazite in masses larger than those of microlite. Zircon has been noted in small crystals and in masses weighing several pounds. Stibnite and galena have been reported, but they are extremely rare and have not been seen by the writer. . . .

Hanover County area.--The zircon-bearing pegmatites of Hanover County form a part of the recently discovered but fairly well-known rutile area of Goochland and Hanover Counties, which lies about 25 miles northwest of Richmond. . . . The principal rock of the region is a gneiss of variable composition, chiefly micaceous (biotite and muscovite) and at times hornblendic, cut by numerous pegmatites, some of which are rutile-bearing, and a variety of basic igneous rocks. Microscopic study of thin sections of the gneiss shows it to conform in composition to an original acidic igneous rock of the granite type. The banded structure is secondary, developed by regional metamorphism. . . .

Zircon has recently been found associated with rutile in the pegmatites near Gouldin in the Hanover portion of the rutile area. . The mineral has been found in irregular fragments and masses up to about 12 pounds in weight. One of the larger masses examined by the writer appears to have been broken from a large crystal of the mineral. Like the other constituents of the pegmatites, every specimen of the zircon studied shows mashing and squeezing from metamorphism. The color is irregular even in the same mass, ranging from reddish-brown through grayish to colorless. Although a chemical analysis of the zircon has not been made, laboratory tests carried out on a number of pieces of the mineral show it to be quite pure. The many pieces of the mineral found on the surface, due to the extensive weathering of the pegmatite bodies, encourage the belief that the mineral is by no means a rare constituent of the dikes in this area and may be found in quantity to be of commercial value.

Lee,[65] in his discussion of hafnium-bearing minerals, stated that a hydrated zircon from Amherst County, first described by Mallett, was under examination.

Washington

Washington has many localities in which the presence of zircon has been noted, chiefly in black sands, old and present beaches, placer gravels, etc.[66] The individual localities, as listed by Day and Richards,[67] are far too numerous for citation here; the counties are as follows: Asotin, Chehalis, Clallam, Clarke, Douglas, Garfield, Okanogan, Pacific, Stevens, Thurston, and Whatcom. San Juan Island is included in the places noted by these authors.

Wisconsin

Gosreau, who investigated methods of ore treatment for the pegmatite deposits in north-central Wisconsin, in Marathon County, approximately 11 miles from the city of Wausau, has given the following report concerning the geology of the district, the character of the zircon, the nature of the tests made upon the ore, and the results deduced:[68]

> The zircon mineral and its associated minerals and rock structures, crystalline shape, and habits, are described by Weidman, from whose report the following descriptive data are taken:[69]
>
> The general geology of the district consists of igneous rocks of diorite, with syenite and pegmatite veins, more or less vertical, intruding the diorite rock masses. These veins are of varying thickness, not definitely known. The pegmatite is coarsely crystalline, a characteristic of these pegmatites. Several varieties of the syenite occur, as well as several mineral phases of the pegmatite. The quartz-pegmatite is composed of quartz and feldspar, and it is the quartz-pegmatite phase that carries the zircon. Other associated minerals are the oxides of cerium, thorium, tantalum, columbium, and yttrium. The syenite carries some small crystals of zircon, of purely mineralogical interest. The diorite does not carry any zircon, according to surface examinations.
>
> The system of pegmatite dikes or veins apparently has a width on the outcrop of about 1,300 feet, and the strike was followed for

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65/ Lee, O. Ivan, Work cited, pp. 24-25.
66/ Morris, H. C., Work cited, pp. 9-13.
67/ Day, David T., and Richards, R. H., Work cited, pp. 1216-1220.
68/ Gosreau, R. C., Recovering Zircon from a Zirconiferous Pegmatite: Eng. Min. Jour.-Press, vol. 119, No. 10, Mch. 7, 1925, pp. 405-406.
69/ Weidman, Samuel, The Geology of North Central Wisconsin: Wisconsin Geol. and Nat. Hist. Survey, Bull. 16, Sci. ser. 4, 1907, 681 pp.

about 2,000 feet. The strike of the outcropping pegmatite veins is northwest-southeast, but no data as to the probable dip are at this time available.

On the surface the pegmatite is much weathered, forming a residual soil from 6 inches to 3 feet thick, in which zircon crystals are found plentifully. Considerable float, in bowlders as large as 10 and 15 inches across, covers the area, these having been thrown in huge piles, so that the land could be farmed. These rock piles are from 75 to 100 feet long, 30 to 40 feet wide, and about 15 feet high. The bowlders furnished the supply of zircon for the milling and chemical treatment of the pegmatite to be described.

An analysis of the aluminous-zircon mineral, given by Weidman, follows:

Table I -- Typical Analyses

	1	2	3
SiO_2	28.87	30.89	31.01
ZrO_2	57.79	60.89	62.12
Al_2O_3	7.80	5.11	4.28
Fe_2O_3	4.47	1.54	1.21
H_2O (red heat)	1.61	1.41	1.76
H_2O (105 deg. C.)	0.43	0.56	0.24
Sp. Gr.	4.28	4.30	4.65
Color	Reddish-brown	Reddish-brown	Pale-yellow
Occurrence	Coarse pegmatite	Fine pegmatite	---

This zircon is high in alumina, accounting for the lower content of ZrO_2, and giving an average oxide content of only 60.3 per cent, against 67.1 per cent for pure mineral.

The apparent quantity of rock available and the fairly high purity of the mineral suggested that mining and concentrating into a product of sufficient richness for commercial uses might under favorable conditions be profitable.

A lot of 3,100 pounds of raw rock was crushed through a jaw crusher to ¼-inch and sampled carefully. The analysis showed 15.9 per cent of ZrO_2 and 26.4 per cent of $ZrSiO_4$. The ¼-inch rock was then ground sufficiently fine to release the zircon from the quartz-feldspar matrix, a minus-35- and plus-65-mesh product being desired, though the crushing and grinding equipment was not controllable to this degree of accuracy.

Water was added to make a 7 to 1 pulp, which was fed slowly to a Wilfley table, using a slight inclination and a long stroke, with plenty of wash water.

. . . The silicate was calculated on a basis of 60.3 per cent oxide in the mineral, and no deduction was made for the alumina content, which was constant. Although the mineral is not pure zircon, the alumina content is not thought prejudicial to its ultimate use. I have attempted to use an aluminate bond and a zirconate bond for zircon, and the natural alumina content may have considerable value.

Concentration ratios: vanner concentrates, 12.5 to 1; vanner tailings, 12.2 to 1.

A pure concentrate rather than high recovery was here sought. Results should be better in a commercial mill. The products containing more than 30 per cent ZrO_2 seem to be suitable for a refractory material, especially the vanner products.

The specific gravities of the minerals involved are: Zircon, 4.0 to 4.7; quartz, 2.3 to 2.7; feldspar, 2.5 to 2.6.

The two vanner products were combined for the chemical treatment. This product averaged 51.1 per cent ZrO_2, thus being 84.8 per cent pure mineral. Considerable iron was also present, which was removed by chemical treatment, to make the following product:

	Per Cent		Per Cent
ZrO_2	57.34	Al_2O_3 (free)	0.65
$ZrSiO_4$	95.00	SiO_2 (free)	4.80
Fe_2O_3	0.30		

This final chemically treated product was dried and ground to 120 mesh. When ground, it had a full white color. Both the final product and the concentration products from this treatment of the raw pegmatite were studied, tested, and found suitable for the following uses:

1. Brick for iron, steel and glass furnace linings.
2. Crucibles, muffles, combustion tubes, insulating tubes.
3. Surface cement for covering other refractory brick, linings, and roofs.
4. Enamel pigment, for iron and steel covering enamels.
5. Added to enamel ware, silica and stoneware, to increase the elasticity and the strength.
6. To prevent devitrification in enameling.
7. Monolithic lining material for walls and hearths, and patch material for electric furnaces making steel, iron, ferroalloys, and brass.
8. Fused in the electric arc and reground, a refractory of high quality would result.
9. A base for making pure white zirconium oxide.
10. An ingredient in magnesia and alumina refractories.

Wyoming

The black sands tested by Day and Richards[10] in the State of Wyoming revealed zircon to be present at Sherman and Keystone, Albany County; Atlantic City and South Pass, Fremont County; Buffalo, Johnson County; Green River, Sweetwater County; and in the Bald Mountain district.

Canada

Very large crystals of zircon, weighing as much as 15 pounds, have been found at Renfrew, Ontario, Canada. The supply is limited, according to Venable.[71] In the description of a new mineral, lyndochite, of the euxenite-polycrase group, from Lyndoch Township, Renfrew County, Ontario, Ellsworth[72] states that among the associate minerals crystal aggregates of zircon or cyrtolite occur the size of a fist.

Marden and Rich[73] report that zircon occurs in crystalline limestone at Grenville (Argenteuil County), Quebec. Zircon syenites exist in Ottawa County, Quebec.[74] In the limestone that surrounds the essexite of Mount Royal, at Montreal, Quebec, zircon is one of the minerals that result from the effects of pneumatolytic contact metamorphism.[75]

Mexico

Zircon is found in the meteoric iron of Toluca, district of Toluca, State of Mexico, Mexico.[76]

West Indies

In a geological survey report upon St. Vincent,[77] the smallest island of the Windward group of the Lesser Antilles, the author states that deposits of black sands, which consist principally of feldspar, mag-

70/ Day, David T., and Richards, R. H., Work cited, p. 1220.
71/ Venable, Francis P., Work cited, p. 13.
72/ Ellsworth, H. V., A New Mineral of the Euxenite-polycrase Group from Lyndoch Township, Renfrew County, Ontario: Am. Mineral., vol. 12, May, 1927, pp. 212-218.
73/ Marden, J. W., and Rich, M. N., Investigations of Zirconium with Special Reference to the Metal and Oxide: Bull. 186, Bureau of Mines, 1921, p. 5.
74/ Fox, Cyril S., Notes on Titanium, Zirconium, Cerium and Thorium: Trans. Min. Geol. Inst. India, vol. 20, pt. 3, Feb., 1926, Calcutta, 1926, p. 265.
75/ Gillson, Joseph L., Work cited, p. 194.
76/ Instituto Geologico de Mexico, Zircon ($ZrSiO_4$): Bol. 40, Catalogo sistematico de especies minerales de Mexico, Secretaria de Industria, Comercio y Trabajo, Mexico, 1923, p. 251.
77/ Earle, Kenneth W., The Geology of St. Vincent and the Neighboring Grenadines: Rept. on the Geol. of St. Vincent and the Neighboring Grenadines, Kingston, 1923, p. 5.

netite, and ferromagnesian minerals, contain a little zircon. A particularly extensive deposit of this kind, about a mile long, occurs between Georgetown and Black Point.

SOUTH AMERICA

Brazil

General (Geological and Mineralogical)

The States of Sao Paulo and Minas Geraes (in the so-called Caldas region) yield the native zirconium oxide, and the States of Bahia, Espirito Santo, and Rio de Janeiro in the beach sands yield the silicate. A deposit of zirconium (about which little is known) is reported in the State of Goyaz, also, north of Minas Geraes.78/

States of Sao Paulo and Minas Geraes.--The ore in the Caldas region, States of Sao Paulo and Minas Geraes, forms (according to a Brazilian Government Report) the following groups:79/

1. Rounded pebbles containing 90 to 93 per cent of zircon in the form of dioxide (ZrO_2), known by the name "favas de zirconio."

2. Massive minerals, as zirkite, light-brown, or caldasite, dark-blue minerals, with a variable percentage of zirconium oxide, 73 per cent in the first mineral and 80-85 per cent in the second one.

3. A good method of enriching being employed, an ore with 80 per cent zirconium oxide (baddeleyite) is obtained.

Three different minerals enter into the composition of "Zirkite" (trade name) and caldasite: baddeleyite, zirconite, and orvillite, a new hydrosilicate of zircon.

The bearing rock is not well known, but the mineral is thought to be in intimate relation with nepheline-syenite rocks that contain eudyalite as a characteristic constituent.80/

This Brazilian occurrence, which is the most extensive deposit of zirconium known, and which can supply industry for many years, is in a plateau region, partly in the State of Minas Geraes and partly in the State of Sao Paulo, approximately 130 miles north of the City of Sao Paulo. Venable describes the region as follows:81/

78/ Cameron, C. R., American consul general, Sao Paulo, Brazil: Trade letter, May 23, 1930, to Philipp Bros. (Inc.), 233 Broadway, New York, Bur. For. and Dom. Com. file.

79/ De Oliveira, Euzebio Paulo, Zircon: Min. Res. Brazil, Servico Geol. e Mineral. do Brazil, Ministerio de Agricultura, Industria e Commercio, Rio de Janeiro, 1930, pp. 27-28.

80/ De Oliveira, Euzebio Paulo, Work cited, pp. 27-28.

81/ Venable, Francis P., Work cited, p. 19.

The mountainous plateau has a main elevation of 3,600 feet. The surface is undulating, presenting differences in level from 300 to 600 feet. The whole area is bounded on all sides by ridges rising abruptly from 600 to 1,200 feet above the general level and forming a roughly elliptical inclosure, with a major axis of approximately 20 miles in length and a minor axis of 15 miles. The predominant rock of the plateau is phonolite.

States of Bahia, Espirito Santo, and Rio de Janeiro.--From Rio de Janeiro to Bahia, on the beaches of the States of Bahia, Espirito Santo, and Rio de Janeiro, zirconium occurs as sand mixed with silicates, principally in monazite sand. "Cusps of the beaches are protected on the north by granite headlands and bordered by Tertiary bluffs cut by various streams and lagoons, which furnish a constant wash of fresh material for the concentrating action of the tides, wave action, and trade winds, which latter are said to play an important part in the final result."[82] This zircon sand is found in large quantities near Victoria (State of Espirito Santo), where it has been mined.

The monazite sand is reported to contain about 63 per cent of the oxide, occurring as $ZrO_2{:}SiO_2$.[83] De Oliveira gave the following analysis in the Government report quoted in a preceding paragraph:[84]

	Per cent
Ilmenite and ferrominerals	14.2
Monazite	61.4
Zirconite	20.7
Quartz.........................	3.7
	100.0

History of the Industry

The most important of the zirconium ores, baddeleyite, was discovered in 1892. It is not known when this ore was first mined. However, according to an American representative in Brazil,[85] the development, expansion, and retrogression of the Brazilian zirconium industry are closely allied with the World War." Before the war, when little interest was manifested by any one in zircon, the ore was shipped to Germany in ballast on homeward-bound vessels and used in the manufacture of gas mantles. In 1914, the German Government, which had been buying increasingly large amounts of sand, placed an order with Mr. Gordon, a producer, for 2,000 tons, at his price. This action of the German Government caused Mr. Gordon to make an investigation, the results of which convinced him that the Germans were obtaining from the sand a metal that formed the basis of extremely efficient

82/ Morris, H. C., Work cited, pp. 9-13.
83/ Cameron, C. R., Work cited.
84/ De Oliveira, Euzebio Paulo, Work cited, pp. 27-28.
85/ Dawson, Claude I., American consul general, Rio de Janeiro, Brazil: Trade letter, Jan. 26, 1929, to Sherlow Chemical Co. (Inc.), 136 Liberty Street, New York, N. Y.

star shells and flares. Also, a few months after the beginning of the war, the British Government discovered the presence of zirconium in the metal of German guns. In 1915 the Brazilian industry was rendered inactive because of the readjustment of markets. From 1916 on exports were made to both England and the United States. As exports and production figures for these years are in almost every instance identical, reference is here made to the table of production in Bureau of Mines Information Circular 6455.

The following table of exports for 1927 and 1928, with countries of destination, is offered by Cameron.[86]

Exports of zirconium minerals, 1927 and 1928

	1927		1928	
	Kilos	Milreis[1]	Kilos	Milreis[2]
Germany	8,000	18:000$	481,723	147:713$
United States	177,000	13:895$	254,000	76:200$
Great Britain	----	----	15,000	6:000$
France	24,950	2:125$	10,200	3:560$
Argentine	----	----	9,100	2:730$
Belgium	49,940	35:885$	----	----
Netherlands	800	320$	----	----
	258,690	70:225$	770,023	236:208$

1/ The Brazilian customs statistics do not distinguish zircon sand from zircon ore, so that there are no available data showing the exports of each.

2/ In 1927, the official rate of exchange was 8$457 and the value in dollars $8,303; in 1928 the rate of exchange was 8$363 and the value in dollars $28,244.

Dawson says[87] that, whereas the readjustment of the market conditions caused a decline in the production during the first few years of the war, high prices tended to expand the industry during 1917 and 1918. The comparatively heavy shipments in 1924 consisted of old stocks liquidated at good prices. Gaulin said[88] that in 1918 zircon sand was produced as a by-product of the monazite industry; that in 1923 the paralyzation of the monazite industry was reflected in the small exports of zircon; and that by 1925 no zircon sand was being mined. A 1929 report[89] is to the effect that small shipments of the oxide ore are being made from the port of Santos to New York and England, at a price of $60 a ton at

86/ Cameron, C. R., Work cited.

87/ Dawson, Claude I., Work cited.

88/ Gaulin, A., American consul general, Rio de Janeiro, Brazil, Zircon and Zirconia in Brazil: Consular Rept., Sept. 5, 1925, Bur. of For. and Dom. Com. file 184835.

89/ Dawson, Claude I., Work cited.

Santos; that the zircon sand mines near Victoria are not being mined, the owners not being willing to resume operations until they are able to obtain an order for at least 250 tons, to be shipped in lots of 20 tons, at a quoted price of 600 milreis, f. o. b. Victoria, to be packed in sacks of 60 kilograms each. Dawson says:90/

> It will be noted that the quotations given above are much higher than the average values per ton, as shown in the official export statistics. Either the official quotations given above are higher than the actual sales prices, or the consular invoice values do not properly indicate the actual value of the merchandise. This is all the more significant as the quotations f. o. b. Victoria given in September, 1925, are substantially the same as those quoted to-day, that is, 600 milreis per ton, while the average value per ton for that year is 274 milreis.

Mining

The deposits in the Caldas region, consisting of outcrops of zirconia, are mined by blasting with dynamite. From these deposits the ore is transported about 20 kilometers by carts, loaded on railway cars at Cascatas, and shipped over the Paulista Railway to Campinas, thence over the Sao Paulo Railway to Santos, a total distance of about 230 miles.91/

Of the beach deposits (States of Bahia, Espirito Santo, and Rio de Janeiro), those near Victoria, State of Espirito Santo, mined as a by-product of the monazite industry, were surface-mined or quasi surface-mined, there being an overburden of about 2 meters of ordinary quartz sand and underlying layers of ilmenite, zircon, and monazite, in the proportions of 33 : 25 : 15, respectively. The product was easily mined. Electromagnetic separators were used to extract the three elements, first the ilmenite, then the zircon, and finally the monazite. The usual run of the mines was 90 per cent zircon. It is reported that the average contents of the zircon sand is 63 per cent oxide (ZrO_2:SiO_2).92/

Owners

Deposits in the Caldas region are controlled by Byington & Co., Largo da Misericordia, 6, Sao Paulo, Brazil (New York office, 165 Broadway), in conjunction with David McKnight, Alameda Barros, 35, Sao Paulo. These deposits of zirconium dioxide ores are near Cascatas and Pocos de Caldas, in Minas Geraes, on the border of the State of Sao Paulo.

Another deposit in this region, at Sao Joao, State of Sao Paulo, was operated during the war. Because of legal differences, its lessors, Messrs. P. S. Nicholson & Co., of Rio de Janeiro, abandoned their interest in the mine; so it is no longer operated.93/

90/ Dawson, Claude I., Work cited.
91/ Cameron, C. R., Work cited.
92/ Dawson, Claude I., Work cited.
93/ Cameron, C. R., Work cited.

The monazite deposits near Victoria, State of Espirito Santo, are owned by Messrs. P. S. Nicholson & Co., Rua Theophilo Ottoni, 45, Rio de Janeiro (S. R. Scott & Co., 31 Pine Street, New York City, American agents), and John Gordon, Rua Visconde Itaborahy, 75, Rio de Janeiro, Brazil.[94]

Cost of Production and Shipping

According to Dawson,[95] no information is available concerning the cost of production of zirconium ore mined in Sao Paulo. The cost of production of zircon sand (near Victoria, State of Espirito Santo) was reported in the latter part of the year 1927 to be as follows:

	Cost per ton in milreis
Cost of production 22$000 per bag of 60 kilograms--16¼ bags per ton--equal to	357$000
Transportation to dock Victoria	44$000
Cost of sacks (at 2$000)	32$500
Export tax	10$000
	443$500

	Cost in dollars
Cost of production in dollars at exchange of 8$500 per dollar	$52.13
Freight to New York	10.00
Miscellaneous shipping charges	2.00
	64.13

British Guiana

Concentrates from placers in the Berbice district, of British Guiana, contain a small proportion of zircon and a high proportion of rutile. Small concessions have been granted to enable further prospecting to be done, but no new developments have been recently recorded.[96]

94/ Dawson, Claude I., Work cited.
95/ Dawson, Claude I., Work cited.
96/ Bulletin of the Imperial Institute (London), Recent Research on Empire Products; Mineral Resources, British Guiana: Vol. 27, No. 1, London, 1929, p. 191.

AFRICA

Gambia

In the report[97] of a geological survey of the Gambia (British West Africa), the description of the cliff and well sections reads, "Heavy minerals are very scanty, except in the grit and conglomerate of the Upper River. They consist chiefly of ilmenite, magnetite and other iron oxides, and zircon, with tourmaline, staurolite, and rutile in much smaller quantities.

In the 21 concentrates obtained by panning material from gutters and streams, in which there appeared to have been some concentration of heavy minerals by nature, magnetite and ilmenite were the most abundant, with zircon second.[98]

Gold Coast

A geological survey bulletin of the Gold Coast (British West Africa)[99] lists zircon with the minerals most frequently found in concentrates obtained by panning the gravels and sands of the country.

Madagascar

A map accompanying a Government report of Madagascar[100] indicates zircon deposits at and to the east of Ambalavara, Beforona, and Vatomandry, which are near the east-central coast; also at Andakatohitsoka and to the east of Fianarantsoa, in the neighborhood of Ampanobe and Ambalavao, the latter places all being in the southeastern part of the island.

Zircon, occurring rather abundantly in Madagascar, is found in pannings, for the most part, in the form of transparent or slightly colored crystals, having a very bright luster. It is found likewise in the many alluvials of the island in the form of crystals, square or octahedral in form. The workable deposits known at the present time are those located in the vicinity of Befrona, of which M. Barbe is the proprietor, and those to the west of Fianarantsoa, of which M. Gauge is the proprietor. The zircon exported from Madagascar assays 60 to 65 per cent; some crystals, however,

97/ Cooper, W. G. G., Report on a Rapid Geological Survey of the Gambia, British West Africa: Gold Coast Geol. Survey Bull. 3, London, 1927, p. 15.

98/ Cooper, W. G. G., Work cited, p. 19.

99/ Kitson, A. E., Minerals in Concentrates; Outlines of the Mineral and Water-Power Resources of the Gold Coast, British West Africa, with Hints on Prospecting: Gold Coast Geol. Survey Bull. 1, 84 Pretoria Road, Streatham, SW., 1923, p. 40.

100/ Bulletin des Mines de Madagascar, À Propos du Zircon: Bull. 12, Dec., 1923, Direction des Mines, Gouvernement General de Madagascar et Dependances, Tananarive, 1923, pp. 217-218.

have shown by analysis 90.7 per cent. The difference, therefore, between this zircon and that from Brazil lies chiefly in mechanical preparation.

Lacroix discusses the workable deposits (in which it is estimated that some thousands of tons are available), mentioned in the preceding paragraph, as follows:[101/]

> M. Barbe reported that he can produce 15 to 30 metric tons of zircon monthly from gold-bearing gravels on the Fanavana, a small branch of the Beforona River. The zircon consists of small brown octahedrons, 1 to 8 millimeters, or 1/24 to 1/3 inch, in diameter.
>
> The Beforona deposit is near the railroad leading to Tanarive.
>
> M. Gaugé discovered, southwest of Solila, zircon crystals, 15 centimeters (6 inches) long, weighing several kilograms. M. Saulnier, a coproprietor, says that the mineral is found over 15,000 hectares to the north and south of Manangantanana, between the Mananbovona and the massif of Ingaro, notably at Mount Ampanobé.
>
> The zircon is found in great abundance. The beds are 4 meters deep. The zircon exists on the surface in a state of concentration; but the lowness of the grade is compensated for by the great extent of the beds. The perfection of form and the sharpness of the angles of the crystals indicate an alluvial deposit. Indications are that the deposit in place is a pegmatite.
>
> Transport from the Mananantanana offers much difficulty.

With respect to the gem grade of zircon, Lacroix says[102/] that zircon crystals resembling those of Ceylon occur in the region of Itrongay, where he found two transparent specimens: one a violet stone, resembling axinite in color, perfectly clear, weighing 13 carats; the other a stone of greater size, but a bit frosty in appearance.

The reddish-orange crystals of zircon collected in the basaltic alluvials of Ankaratra only occasionally furnish usable stones.

Namaqualand

Rogers,[103/] in giving his results of the microscopic study of 200 rocks of Namaqualand (German Southwest Africa) reports zircon in the norite, in the augite-diorite, in the mica diorite, and in the granite rocks.

101/ Lacroix, A., Les Industries minérales non métallifères à Madagascar: Paris, 1920, pp. 53-58.

102/ Lacroix, A., Les Pierres de Madagascar; Gemmes pierres d'ornementation, pierres d'industrie, Paris, 1921, p. 36.

103/ Rogers, A. W., The Nature of the Copper Deposits of Little Namaqualand: Anniversary Address by the President; Minutes of the Proceedings of the Geol. Soc. of South Africa: Proc. Geol. Soc. South Africa, Jan. to Dec., 1916, Johannesburg, 1917, p. 22.

Nigeria

Northern Nigeria.--Prof. Wyndham Dunstan has made a number of surveys of the minerals of Northern Nigeria. The surveys were made of the concentrates of river beds for the purpose of detecting monazite and thoria chiefly, but the presence of zircon is reported in many of the localities studied.104/ The reports are far too detailed for even a digest here.

Southern Nigeria.--Under the heading of "Concentrates," zircon is mentioned as one of the stones of the river beds in a number of reports made by Professor Dunstan upon mineral surveys made of Southern Nigeria. (The concentrates were examined for their monazite and thoria content chiefly.) These reports, beginning in 1906, cover a number of years and are rather detailed.105/

Nyasaland

Three reports by Dunstan106/ record the occurrence of zircon in the Protectorate of Nyasaland (British Central Africa).

Rhodesia

General.--With respect to zirconium-bearing ores in Rhodesia in general, Zealley has reported as follows:107/

> Baddeleyite is not known to occur in Rhodesia. Zircon is of such widespread occurrence that if granite soil in any part of Rhodesia be panned, a tail of minute gray or reddish-brown zircon crystals will be noted; whilst pannings of the sand in the beds and banks of granite spruits will afford a much larger concentrate. There are such enormous tracts of granite in Rhodesia that the mineral could be recovered in large quantities. Two occurrences examined by the writer in 1917 appear to be worth commercial investigation. These are at Hillside, Bulawayo, and in the stream courses around Rusape. The Hillside syenite contains much more zircon than most granite rocks in Rhodesia; moreover the mineral appears in much stouter crystals--they may be as big as 1/8 inch; they are lustrous and generally reddish-brown, but partly colorless like quartz.

104/ Dunstan, Wyndham, Reports on the Mineral Survey of Northern Nigeria: Colonial Repts., Misc., 1904 to 1907, inclusive, Imperial Institute, London.

105/ Dunstan, Wyndham, Reports on the Mineral Survey of Southern Nigeria; Colonial Repts. Misc., 1903 to 1914, inclusive, Imperial Inst., London.

106/ Dunstan, Wyndham R., Report on the Results of the Mineral Survey: Colonial Repts., Misc., Nos. 48, 60, and 80, 1906-7, 1907-8, 1908-9, London, 1908, 1909, and 1911.

107/ Zealley, A. E. V., Zirconium Ore: Rept. Rhodesian Resources Comm. 1921, Bulawayo, 1921, pp. 130-131.

Specimens and information fro Rusape indicate that zircon is an abundant constituent of granite sands and spruits in that region. The concentrates contain corundum, magnetite, ilmenite, scheelite, epidote, haematite, hornblende, tourmaline, sphene, rutile, and several other minerals, all in small quantities except the iron minerals, which could be removed by magnetic concentration. * * *

Southern Rhodesia.--Small quantities of ilmenite, zircon, and rutile are present in the concentrates of ore from Eileen Alannah mine, as ascertained by analyses. The mine Eileen Alannah is northeast of Gatooma, in the Eiffel Flats group of mines. The general description of the region in which this mine is located is that it is east of Gatooma and that it lies east of the Bulawayo-Salisbury Railway line, which runs in a northeast direction from Umsweswe Siding, through Gatooma, to Hartley.108/

Zircon is mentioned in a discussion of the petrography of the greenstone schists (under which term are grouped a number of metamorphic rocks, composed largely of hornblende) of the Enterprise Mineral Belt (Enterprise is a popular name for the farming and mining district situated directly east of Salisbury). The mineral belt is a strip of country extending east from Salisbury for about 35 miles, and measuring approximately 8 miles from north to south. The mineral belt forms part of the Salisbury mining district.

A sample of epidiorite with the massive type of hornblende from a hill 3/4 mile west-southwest of Chishawasha Mission contains regular inclusions of quartz, as well as of apatite and zircon, the last-named being surrounded by pleochroic haloes. On Father Hartman's farm the proportion of zircon is epidiorite was high.109/

In a description of the geology of the diamond-bearing gravels of the Somabula Forest, under the heading "Karroo System," under the subheading "Mineralogy of the Concentrates," is the following:

> Zircon is an uncommon constituent of the concentrates. It forms pale chocolate-colored crystals, with pyramidal cleavage. Zircon commonly occurs in Rhodesian granites, notably in the neighborhood of Rusapi.

Somabula Forest forms a narrow belt 1 to 10 miles wide, stretching in a northwesterly direction for a distance of 70 miles and more, its range being controlled by the distribution of the Somabula beds. The part of the Somabula Forest dealt with in the report lies between 19° 20' and

108/ Zealley, A. F. V., Preliminary Report on the Geology of the District East of Gatooma: Southern Rhodesia Geol. Survey Bull. 1, Bulawayo, 1913, p. 19.

109/ Maufe, H. B., The Geology of the Enterprise Mineral Belt: South Africa Geol. Soc. Bull. 7, Salisbury, 1920, pp. 14 and 17.

19° 50' south latitude and between 29° 30' and 29° 50' east longitude. (For a fuller description, see pages 9 and 10 of the article referred to in thd footnote.)110/

Senegal

Black, fine, and heavy sands on the coast of the Senegal (French West Africa), called ilmenite sands, and generally sold for their titanium content, in some places contain more zirconium than titanium, the average in one concession being 29 per cent zirconium and 28 per cent titanium.

These sand deposits are known to occur on the ocean beach in the vicinity of Dakar (at Rufisque, 15 kilometers across the bay from that city, and at Joal, about 100 kilometers south of Dakar); south of the village of Diakonar, in the vicinity of Bargny Guedje, Diogue, Thies, and St. Louis; and on the borders of Matam and Bakel.

Sierre Leone

A Government report of the Protectorate and Colony of Sierra Leone (British West Africa)111/ states that a few small brilliant crystals of zircon, as well as corundum and sapphire, were found in the gravels of the Little Scarcies River, for a few miles below the conjunction of this river with the Mabole River. The principal minerals of these sands, however, were rutile and ilmenite.

Uganda

The Uganda Geological Survey reports112/ that zircon occurs widely but is abundant in only one area in the Uganda Protectorate (British East Africa).

As the result of a geological investigation of a portion of the northern spur of Ruwenzori, principally on the western side of the mountain (from the Karimi River in a southwesterly direction to the Lamia River), but also in a small area on the eastern side of the mountain (from the Mpanga River in a southwesterly direction to the Nyabuswa River), and also several miles out from the mountain foot in Bwamba Province, the report was that the gravels from nearly all the streams and valleys yield very large concentrates, composed of coarse subangular magnetite and haematite, with a smaller amount of garnet, zircon, and other minerals.113/

110/ Macgregor, A. M., The Geology of the Diamond-Bearing Gravels of the Somabula Forest: Southern Rhodesia Geol. Soc. Bull. 8, Salisbury, 1921, p. 26.

111/ Report of the Geological Department for Part of the Year 1927 and for the Year 1928, Geological Department of the Colony and Protectorate of Sierra Leone, Freetown, 1929, pp. 14-15.

112/ Uganda Geological Survey Annual Reports, 1920 and 1925.

113/ Combe, A. D., Field Work--Northern Spur of Ruwenzori and Part of Bwamba, May 29 to August 19: Ann. Rept. Geol. Survey Dept., 1924, Uganda Protectorate, Entebbe, 1925, p. 6.

Many of the concentrates from streams in the Province of Karamoja show grains that are easily recognizable as zircon. The grains are bright in luster and are either colorless or pinkish or pale-brown in color. Many of them are crystalline in form.114/

Hirst reports115/ that three distinct habits of zircon occur in the valley of the Kafu River, and that apart from gold it stands second in abundance among the minerals in the normal concentrates.

Union of South Africa

General.--Lindgren, in a discussion of diamonds in his work on mineral deposits,116/ lists zircon with other minerals found in the blue ground of the Kimberley pipe in the Kimberley diamond field. The blue ground is the still unoxidized layer under the decomposed serpentine, or yellow ground, which is near the surface.

The Kimberley diamond field in South Africa lies in the northern part of Cape Colony and the adjacent part of the Orange Colony. Another district centers at Jägerfontein, in the Orange Colony; still another centers at the Premier mine, near Pretoria, in the Transvaal.

Cape of Good Hope.--In wash from molteno beds (as shown by a specimen from Molteno) the diamonds are accompanied by red garnet, rutile, and numerous other minerals, all of a size similar to that of the diamond, the zircons, anatases, and rutiles being fresh and sharp. Zircon is plentiful in little prisms, terminated at both ends, and having very strong black borders.117/

In a description of the geology of the districts of Prieska, Kenhardt, and Carnarvon, Rogers and Du Toit,118/ in discussing the Marydale beds of the Kheis Series, mention zircon in a gray schistose rock formed of quartzite, biotite, sillimanite, magnetite, and a little feldspar, on Lower Rooi Puts, in the Rooi Puts belt. Grains of zircon were observed in the interstitial matter, together with allite, microline,

114/ Simmons, W. C., Laboratory Work; Rare Earth Minerals: Ann. Rept. Geol. Survey Dept., 1926, Uganda Protectorate, Entebbe, 1927, p. 35.

115/ Hirst, T., Summary of Work Carried out in the Kafu Valley in 1926: Ann. Rept. Geol. Survey Dept., 1926, Uganda Protectorate, Entebbe, 1927, p. 17.

116/ Lindgren, Waldemar, Mineral Deposits: 2d ed., McGraw-Hill Book Co. (Inc.), New York, 1919, pp. 787-788.

117/ Schwarz, Ernest H. L., Diamonds from the Molteno Beds: Trans. Geol. Soc. South Africa, vol. 19, Johannesburg, 1916, pp. 33-35.

118/ Rogers, A. W., and Du Toit, A. L., Geologic Survey of Parts of Kenhardt, Prieska, and Carnarvon: 14th Ann. Rept. Geol. Comm., 1909, Cape of Good Hope Dept. of Agriculture, Cape Town, 1910, pp. 9, 27, and 42.

quartz, colorless garnet, muscovite, biotite, and magnetite, in the mass of pebbly grit and quartzite lying in the gneiss, partly on Piet Eksteen's Kuil, and partly on Piet Rooi's Puts.119/

The region described extends from the Orange River below the Prieska boundary to Upington and southwards to the vicinity of Carnarvon. The Hartebeest and Zak Rivers form the western limit from Staan-still-en-afspring on the Zak to Vyf Beker on the Hartebeest. From the farm (just named) on the Zak the boundary runs through Verneuk Pan past Zwart Kop, Hartog's Kloop, Boter Leegte, and Meintjes Kloop to the neighborhood of Carnarvon, thence to Victoria West and Van Wyks Vlei, across the Kaaien Bult to Nels Poortje, along the southwestern flank of the Kaaien Hills to Karree Leegte and across the hills to the Orange River on Uit Draai.

In a description of a petrological examination of the volcanic rocks of Matatiele Division, East Griqualand, Schwarz reports the presence of zircon, as follows:120/

1. In red bed between the lavas, top of Drakensberg, on Eyrie.
2. In siliceous rock interbedded in lava, top of Drakensberg.
3. In volcanic breccia, neck on the farm N'Quatsha.
4. In volcanic neck, filled with agglomerate, on the farm N'Quatsha.
5. In the cave sandstone from N'Quatsha's Nek road.

N'Quatsha and Eyrie are two of the volcanoes the lava flows from which form the crest of the Drakensberg Range.

The following information has been supplied by the inspector of mines, Pretoria district, with reference to the availability in the Union of certain minerals for use in the manufacture of cheap jewelry:121/ Clear zircons up to several hundred carats in weight occur in large quantities on the surface of the Zibarc mine, in the Carnarvon district of the Cape Province. The quantity is so great that they probably could be produced at the price of 2s. or 3s. a carat.

Transvaal.--In a discussion of the corundum fields in the northern and southern Transvaal, under the heading "Mineralogy of the Deposits," Hall says: "Rutile and zircon were found only in thin sections, the former in small regular deep-brown crystals in the ruby-disthene rocks from Mashishimala, Malelane, etc., the latter as inclusions in biotite of the Palmietfontein corundum reef."122/

119/ Rogers, A. W., and Du Toit, A. L., Work cited, p. 42.
120/ Schwarz, Ernest H. L., Petrological Examination of the Volcanic Rocks of Matatiele, Griqualand, East: Ann. Rept. Geol. Comm. 1903, Cape of Good Hope Dept. Agriculture, Cape Town, 1903, pp. 94-96.
121/ Trevor, T. G., The Common Gem Stones of the Union: South African Min. Jour., vol. 27, Johannesburg, Ja. 26, 1918, p. 486. (No. 1374.)
122/ Hall, A. L., Corundum in the Southern and Eastern Transvaal: Geol. Survey Mem. 15, Union of South Africa Dept. of Mines and Industries, Pretoria, 1920, pp. 38-39 and 136.

Special mention was made by the same author[123] of minute inclusions of zircon carried in plagioclase (andesine), a deep-greenish biotite in the Palmietfontein "reef" in Palmietfontein No. 374 corundum mine, 12 miles southwest of Louis Trichardt, Zoutpansberg district (northern Transvaal), in the plateau region west of the Pietersburg-Messina Railway.

Zircon is a fairly constant constituent of the <u>banket</u>, a South African mining term for beds of auriferous conglomerate, chiefly occurring in the Witwatersrand goldfields. (The Witwatersrand is a chain of mountains in the Transvaal from which 42 per cent of the world's gold is produced.) Zircon can be observed in about 20 per cent of the microscopic sections of the rock. Frequently the crystals exhibit prismatic and pyramidal faces. Others are irregular in form and fragmentary in appearance. The crystals, too, occasionally show rounding of the edges. The average length of the crystals is 0.2 millimeter, and their thickness is about two-thirds or one-half of the length. They are transparent and vary in color from almost colorless to pink, sometimes having a brownish tint. The crystals are conspicuously zonal in character. Zircons are found in the quartzite also, associated with the banket, though apparently in much less quantity.[124]

In a description of the Salt Pan on the Farm Zoutpan, No. 467, north-northwest of Pretoria (the Pan is a remarkable, flat-bottomed, crater-like depression within circular rims of red granite that rises in the form of a low range of bush-clad hills, about 25 miles northwest of Pretoria), Wagner says[125], that underneath each trona bed (of which there are sometimes five) there is a layer of mud or clay, in which small grains of zircon and rutile were noted. The crude trona contains appreciable quantities of organic matter, sodium chloride, and insoluble material.

At Gibeon (Gibeon district) and Mukorub (Berseba district), bright grains of zircon occur in the Gibeon and Mukorub pipes, where they have been mistaken for diamonds.[126]

Zanzibar

In the Zanzibar Protectorate (British East Africa), in the sands of varying grades and nature in the different strata of the Pemba Series and the upper portions of the Zanzibar Series, which contain high percentages of iron, zircon, ilmenite, etc., are frequently present.

123/ Hall, A. L., Work cited, p. 89.

124/ Young, R. B., Further Notes on the Auriferous Conglomerates of the Witwatersrand, with a Description of the Origin of the Gold: Trans. Geol. Soc. South Africa, vol. 12, Johannesburg, 1909, pp. 84-85.

125/ Wagner, Percy Albert, Some Problems in South African Geology: Proc. Geol. Soc. South Africa, 1917, Johannesburg, 1918, pp. 30-33.

126/ Wagner, Percy Albert, The Geologic and Mineral Industry of South West Africa: Geol. Survey Mem. 7, Mines Department of Union of South Africa, Pretoria, 1916, p. 72.

According to an official report, so far the zircon has been considered only as prejudicial to the economic use of the sands as glass or molding sands.127/

ASIA

Ceylon

An American representative in Ceylon,128/ after consultation with the Government mineralogist and a local gem expert, reported that although zirconium ore is found in large quantities in the island it has never been exported. Zircon is found in commercial quantities in the black sands of the sea beach, but it would not be practicable to collect the mineral on a large scale. No mention of exports of zirconium ore is made in the customs returns of Ceylon. Another representative129/ states that gem stones are shipped to Europe and America, the best stones being exported and the inferior ones being sold locally and in India.

Most of the literature concerning the zirconium of Ceylon relates to the gem grade of zircon, which is to be discussed at some length in a forthcoming information circular upon the zircon. In brief, the streams flowing through the Balangoda, Rackwana, and Ratnapura districts, in the central part of southern Ceylon (half-way between the city of Kandy and the southern shore of the island), yield the principal supplies of the gem stones.130/ At Matara, near Dondra Head, at the extreme point of the island, zircon is found in colorless individuals that are sufficiently large to be cut and employed in jewelry, being known in trade as Matara diamonds.131/

Although the southwestern part of Ceylon, where zircon is a very common accessory constituent in the rocks along the coast, is the principal zircon-bearing region, the mineral is very common in all the gneisses of the crystalline area, and gneisses underlie a very large part of the island. The zircon appears in thin sections of the rock in the form of small rounded individuals or in minute prisms, having rounded edges.132/

Fox reported the existence of baddeleyite at Rakwana, Ceylon.133/

127/ Stockley, G. M., Report on the Geology of Zanzibar Protectorate: Govt. Rept., March, 1928, pp. 102-103.

128/ Turner, Mason, American vice consul, Colombo, Ceylon: Trade letter, April 27, 1925, Bur. For. and Dom. Com. foreign file.

129/ Vance, Marshall M., American consul, Colombo, Ceylon, Report on Mineral Deposits and Industries in Ceylon for the Year Ended December 31, 1921: Consular Rept., Aug. 2, 1922, Bureau of Mines foreign file 4138.

130/ Adams, Frank Dawson, A Visit to the Gem Districts of Ceylon and Burma: Ann. Rept. Smithsonian Inst., 1926, 1927, pp. 297-318.

131/ Adams, Frank Dawson, The Geology of Ceylon; Geology: Canadian Jour. Res., vol. 1, No. 5, Nat'l. Research Council of Canada, Ottawa, Nov., 1929, p. 461.

132/ Adams, Frank Dawson, The Geology of Ceylon; Minerals of Economic Value, Gems: Canadian Jour. Res., vol. 1, No. 6, Nat'l. Research Council, Ottawa, Dec., 1929, p. 508, (Also, The Geology of Ceylon; The Quartzose Biotite Gneisses of the Ordinary Type, vol. 1, No. 6, p. 467.)

133/ Fox, Cyril S., Work cited, p. 265.

A mineral survey of Ceylon was instituted in 1902, at the suggestion of the Director of the Imperial Institute, to examine the existing deposits of minerals of economic importance and to search for new deposits, in order to pave the way for commercial development. The results of the survey are summarized in the following paragraph.134/

1. Shore deposits of the West and South Coasts.
   (1) Shore deposits from Colombo northwards.
   From Colombo to Dutch Bay, garnet, zircon, and rutile were the common associates of the monazite in the nambu (a natural concentrate of heavy minerals; also employed to denote the residue of heavy minerals left after washing the gem gravel and removing valuable gems).
   From Puttalam northwards zircon was found to be rather less common.
   (2) Coast belt between Colombo and Hambantota.
   A sample of concentrate from the fourth bay south of Bentota River (examined by the Imperial Institute), consisting chiefly of monazite and ilmenite, yielded some zircon and rutile. A concentrate from Kaikawala, consisting chiefly of monazite, yielded small amounts of zircon, with other minerals.

2. Alluvial deposits in the Ratnapura district.
   (1) Kalu Ganga Valley.
   Seventeen samples of concentrates from sands obtained from drilling in the bed of the Kalu Ganga. Although ilmenite and garnet were the chief stones, some zircon (with other minerals) was found.
   (2) Gravel of the We Ganga.
   Fourteen concentrates obtained by boring in the river beds and adjacent flats, while consisting chiefly of ilmenite, carried zircon as well as other minerals. Dredging in the We Ganga showed the same minerals in practically the same proportions.
   (3) Other deposits.
   A promising nambu, made up of zircon, garnet, spinel, tourmaline, and corundum, found in a drill bed sunk in Niwitigala village, in the Dela district. Gravel is fairly extensive here.

134/ Bulletin of the Imperial Institute (London), Reports of Recent Investigations at the Imperial Institute; Recent Work on Monazite and Other Thorium Minerals in Ceylon: Vol. 14, No. 3, July-Sept., 1916, London, 1916, pp. 321-369.

3. Gravels of the Kelani Ganga.
(1) Between Madagoda and Pugoda.
Sixteen concentrates examined from the Pugoda Ganga bore holes: ilmenite predominant mineral; zircon and others in comparatively small amounts.
(2) Between Pugoda and Malwana.
Nine concentrates. Practically as above.

4. Gravels of the Sitawaka Ganga.
Twenty-seven concentrates examined by the Imperial Institute. Ilmenite predominant. Zircon included with other minerals present.

5. Nuwara Eliya district.
(1) Nuwara Eliya Plains.
The quartz gravel yielded a little black tourmaline and zircon crystals.
(2) Moon Plains.
A remarkable nambu, rich in monazite, opaque corundum, zircon, and zenotime, with a few gem stones, was obtained from a small flat on the Rifle-range Stream, near the head of its gorge. The dike rock is an orthoclase pegmatite. Samples (taken at one time) showed 12 grams of zircon a metric ton. Zircon crystals can be picked out of the decomposed rock by crumbling up the lumps of kaolin. In the gravels yellow transparent zircons are common.
Samples taken at another time showed that the concentrate from pegmatite consisted chiefly of zircon, with some biotite, ilmenite, and garnet.
The Magoda Oya gravels yield a nambu rich in corundum and chrysoberyl, with zircon and black tourmaline. Concentrate from the vicinity of Nuwara Eliya showed small amounts of zircon.
(3) Elk Plains.
The zircon found here (not in valuable amounts) differs from that in the Moon Plains pegmatite in having a brownish-pink color.
(4) Horton Plains.
As on the Elk Plains, pink zircon accompanies the monazite.

6. Thorianite and thorite deposits in the Bambarabotuwa, Denawak Ganga, and Walawe Ganga districts.
(To revive interest in the production of thorianite, headmen were informed in 1915 of the increase in the value of the mineral, and efforts were made to examine samples.)
(1) Bambarabotuwa district.
Prospecting on the Kuda Pandi Oya showed thorianite in all the veins as the principal accessory mineral, with

ilmenite next in importance, and small quantities of thorite, monazite, and zircon.

(2) Denawak Ganga district.

Dredging in the Denawak Ganga along the steeper courses of the stream yielded well-rolled fragments of gem minerals, chiefly zircon and spinel, with a trace of thorianite, at a depth of 4 to 5 feet.

(3) Walawe Ganga district.

A number of mineral specimens from Walaweduwa, examined by the Institute, proved to be zirkelite, containing 36.2 per cent zirconia, thoria 17.4; titanium dioxide 24.6; and lime 6.7.

(4) Thorianite at Niralgama.

Niralgama, 5 miles south of Ratnapura. In addition to thorianite (predominant) were found thorite, fergusonite, rutile and zircon.

(5) Prospecting for thorium minerals in the Yalkumbura district.

An outcrop of pegmatite, with large phlogopite crystals, occurs near the head of the dry dola, and pannings from the surface gave abundance of fine monazite. Material from the body of the vein gave abundance of zircon. Specimens from Yalkumbura examined by the Imperial Institute showed zircon from pegmatite. A concentrate consisting almost wholly of zircon, contained some quartz, monazite, and small amounts of rutile, ilmenite, hornblende, and magnesite. The monazite amounted to about 4 per cent. Zircon was present also in monazite from gullies and in the monazite from pegmatites; the thorite from gullies consisted chiefly of zircon and spinel, with some thorite and rutile--15.66 per cent of thoria.

India

Fox says[135] that zircon is a common accessory mineral in many granitic and gneissose rocks in India. Large crystals are sometimes found in certain pegmatites that contain rare-earth minerals. Zircon concentrates, however, are most commonly found in the river sands and in the beach sands in the neighborhood of zircon-bearing rocks.

Travancore.--The principal deposit of zircon in India and the only one ranking as a commercial source of the mineral is in Travancore. The Travancore beach deposits, which are now exploited chiefly for their ilmenite, contain appreciable quantities of zircon, as well as monazite, the zircon carrying 62 to 63 per cent of zirconium oxide, the ilmenite 50 to 53 per cent titanium oxide, and the monazite 8.50 to 9.20 per cent thorium oxide. These deposits are renewed several times a year by the sea, along an extent of 79 miles.

135/ Fox, Cyril S., Work cited, pp. 257-258.

The output of zircon, obtained as a concurrent product in the collection of ilmenite and monazite in Travancore State, decreased from 1,465 tons, valued at £8,129, in 1927, to 885.2 tons, valued at £4,267, in 1928, in spite of a conspicuous increase in the production of ilmenite.136/

The company that is mining the Travancore deposits is The Travancore Minerals Co. (Ltd.), 52 Queen Victoria Street, London, England, which has a local office at Colachel, South Travancore, India.

For a description of the Travancore beach sands (written especially with reference to their monazite content), see the footnote reference.137/

The following notes are, for the most part, taken from transactions of the Geological Institute of India.138/

Bihar and Orissa.--Large clusters of dark-brown crystals of zircon have been found in the pegmatite of Abraki Pahar, in the Gaya district. In that locality it is associated with pitchblende.139/

Burma.--Appreciable percentages of zircon crystals were found in an ilmenite sand, containing monazite, from Upper Burma. It is understood that the sands are attractive for their monazite content. The zircon and ilmenite would be useful by-products.

Chitral.--Zircons have been found in the magnetite-ilmenite sands of the streams of Chitral. The quantity and quality of the material are not very attractive, and the localities in which it is found are inaccessible.

Madras.--Madras State is by far the most important area in India with respect to the rarer minerals, titanium, zirconium, cerium, and thorium. Zircon has been located in the nepheline syenites near Kangayam, in the Coimbatore district, in the pegmatites at Kadavur of the Trichinopoly district, and in the Seitur graphite mine at Ramnad. A hydrated form of zircon (resembling cyrtolite), containing a small percentage of uranium, is associated with samarskite in the Nellore district. Bohm140/ reports zircon in the Province of Circars, on the eastern coast of the Madras Presidency.

136/ Records of the Geological Survey of India, Zircon: Vol. 62, pt. 3, 1929, Calcutta, 1929, p. 332.
137/ Tipper, G. H., The Monazite Sands of Travancore: Rec. Geol. Survey of India, vol. 44, pt. 3, Nov., 1914, pp. 186-196.
138/ Fox, Cyril S., Work cited, pp. 257-258.
139/ Records of the Geological Survey of India, Pitchblende: vol. 44, pt. 1, May, 1914, pp. 24-26.
140/ Bohm, C. Richard, Das Aufschliessen der wichtigsten Mineralien; Zirkon: Die Darstellung der seltenen Erden, Leipzig, 1905, pp. 108 and 116.

With respect to the gem grade of zircon, Brown[141] reports that fine crystals come from an unknown locality in the Vizagapatam district of Madras. He reports also that red and grayish-white crystals, transparent in part, occur in pegmatite veins at Appiyode in the Eraniel taluk of Travancore. The zircon gem stone is found also, in minor quantities, in the ruby mines of Upper Burma, the principal ruby-bearing districts being Mandalay, Mytkynia, and the one about Mogok, which includes Kathe. The Burma Ruby Mines (Ltd.), whose chief operations are at Kathe, has workings at Enjouk, Bigom, Nayasen, and other points. The zircons are found in the byon, which corresponds to the illam of Ceylon, and which occupies a position identical with that of the gold gravels in many alluvial regions.

Japan

The following facts with respect to the zircon and other zirconium-bearings ores of Japan are taken from reports by Shibata and Sato.[142 & 143] The matter is presented by ores, as one of them, naëgite, is found only in Japan, and oyamalite very closely resembles it.

Zircon and xenotime.--Zircon and xenotime occur in small tetragonal crystals at Ishikawa, Iwaki Province. The color of the former is reddish-brown or brown; the latter is brown or reddish-green. Careful investigation disclosed the fact that these minerals are almost always found associated or intermingled with each other; and that, moreover, sometimes zenotime occurs inclosing zircon in a parallel position.

Analysis of this zircon gave ZrO_2 58.71 per cent and SiO_2 32.40 per cent. The total analysis for zenotime gave ZrO_2 19.84 and SiO_2 12.49 per cent.

Minute and well-crystallized zircon is found in the sand of the Keelung-gawa, Formosa. Large crystals of this mineral occur in a graphite deposit at Jido, Heian-hokudo, Korea. Two specimens of Korean sand, from Jun-an and Shokusan, have been studied. The sand from the latter place was reported to contain 1.05 per cent of zirconium oxide (ZrO_2).

Naëgite.--This special Japanese mineral has been found nowhere except at Naegi (Mino Province) up to the present time. It is a brownish-green mineral, having a resinous luster. Its crystal form resembles very much that of zircon; its specific gravity is 4.1; and its hardness is 7.5.

141/ Brown, John Coggin, India's Mineral Wealth: India of To-day, London, 1923, pp. 104-105.

142/ Shibata, Yuji, The Chemical Investigation of Japanese Minerals Containing Rarer Elements: Proc. 3d Pan-Pacific Sci. Cong., Oct. 30 to Nov. 11, 1926, vol. 1, Nat'l. Research Council, Tokyo, 1928, pp. 852-865.

143/ Sato, Denzo, Some Minerals Containing Rarer Elements: Proc. 3d Pan-Pacific Sci. Cong., Oct. 30 to Nov. 11, 1926, vol. 1, Nat'l Research Council, Tokyo, 1928, pp. 865-866.

Analyses show (1) ZrO_2, 53.03, and SiO_2, 29.55; and (2) ZrO_2, 51.24, and SiO_2, 30.48.

Recently Hevesy reported that naëgite contains about 3.5 per cent hafnium.

Naëgite has been described as a variety of zircon (radioactive zircon), but it has also been considered to be an isomorphous mixture of ZrO_2, ThO_2, UO_2, and SiO_2.

Hagatalite.--The mineral hagatalite resembles zircon and occurs commonly in small crystals, 1 millimeter to 5 millimeters in diameter, imbedded in biotite. Its color is yellowish-green or green; its specific gravity is 4.4; and its hardness is about 7.5. Analysis gave ZrO_2, 42.0 and SiO_2, 29.7. "From the results of analysis it is concluded that this mineral, like naegite, may be a variety of zircon, which contains rare earths and rare acid earths. Hagatalite, however, is clearly distinguishable from naëgite by its content of zirconium and also by its general appearance. Hagatalite occurs imbedded in the pegmatite of Hagata-mura.

Oyamalite.--Oyamalite from Oyama, Iyo Province, closely resembles naëgite. It occurs imbedded in feldspar, is of a dark green or brown color, has a specific gravity of 4.1 and a hardness of 7.5. Analysis showed 40.9 of ZrO_2 and 25.7 of SiO_2. Evidently oyamalite is a variety of zircon but differs from the latter in that it contains phosphoric acid, whereas zircon seldom does.

Siam

The gem grade of Siamese zircon is discussed in greater detail in a forthcoming information circular upon the semiprecious stone. Eppler refers to blue zircon from Chantaboon, Siam.144/ An American representative in Siam145/ says that, although the quantity of zircon stones produced is not recorded, it is known that they are mined in the region of Chandeburi and the territory along the Mekong River, which forms part of the boundary between Siam and French Indo-China.

The zircons are mined at a depth of 5 to 10 feet below the surface in an alluvial deposit. Single pieces weighing 525 carats have been found.

144/ Eppler, W. Fr., Über das optische Verhalten und Zustandsänderungen des Zirkons: Forts. Mineral., Kristall. und Petrog. vol. 1, Jena, Germany, 1927, pp. 302-303.

145/ Albrecht, Charles H., American consul, Bangkok, Siam, Zircons: Consular Rept., July 17, 1926, Bureau of Mines foreign file No. 9203.

AUSTRALASIA

Australia

The gem grade of zircon is reported to occur in New South Wales, Queensland, Tasmania, and South Australia. Government reports of Queensland and New South Wales have included figures of production for the gems, although output has not at any time been very large. Zircon sands also are found in several localities.

New South Wales.--Zircon, the only zirconium mineral known to occur in New South Wales, is a minor accessory mineral in the more acid igneous rocks, such as granite and pegmatites, in river gravels and in beach deposits. It is found in small prismatic crystals, which are a little harder than quartz but not so hard as topaz, and which have a very brilliant luster.146/

Carne says147/ that the auriferous sands of the Esk River and Jerusalem Creek, in the Parish of Esk County, carrying ilmenite, gold, platinum and platinum metals, consist primarily of zircon. They contain some tin also.

A Queensland Government report of 1907148/ reported a proposed development of the sands along the beach of New South Wales, where gold, platinum, tin, monazite, zircon, and other minerals were known to exist. The area to be exploited was 13 miles long by 10 chains wide, paralleling the beach, beginning about 1 mile south of the Evans (or Little) River, approximately 20 miles south of Ballina, at the mouth of the Richmond River, and extending northward to what is known as McAuley's Lead, to within approximately 10 miles of Clarence River. The same publication reported valuable sand deposits about 80 miles north of the beach just mentioned, near Currumbin Creek. Small-scale treatment of the sands had been carried on between Currumbin Creek and Coolangatta.

Western Australia.--Dr. Garcés149/ says that Phipson (Compt. rend. Acad. Sci., vol. 64, pp. 87-88), from a mineralogical study made

146/ Raggatt, H. G., Chromium, Cobalt, Nickel, Zirconium, Titanium, Thorium, Cerium: New South Wales Geol. Survey Bull. 13, Dept. of Mines, Sydney, 1925, p. 14.
147/ Carne, J. E., The Auriferous Sands of the Esk River and Jerusalem Creek in the Parish of Esk County, Richmond, New South Wales: New South Wales Geol. Survey, Vol. 5, 1897, pp. 71-86.
148/ Queensland Government Mining Journal, Beach Mining in New South Wales: Vol. 8, No. 83, Queensland Dept. of Mines, Brisbane, April 15, 1907, p. 175.
149/ Garcés, V. Soriano, Arena Circonífera de Vigo: Trabajos del Museo de Ciencias Naturales de Barcelona, vol. 9, No. 2, Publicaciones de la Junta de Ciencias Naturales de Barcelona, Barcelona, Aug. 13, 1928, p. 9.

with the microscope of the metalliferous sands of Fremantle, Western Australia, distinguished silicate of zircon, together with quartz, topaz, apatite, and some diamonds.

Simpson says[150] that zircon has been shown to be present in many crystalline rocks in this State, where it can be detected in small, well-formed, colorless or smoky crystals by panning any of the clays (that is, of Bellevue, Belmont, and Mujar) or the river or beach sands, particularly the sands that show through the presence of black iron compounds that they have been concentrated naturally. Simpson lists the following sands where zircon has been found: South Perth, black sand, estuary beach; Cottesloe, white and black sand, sea beach; Koombana Bay, black sand, sea beach; Nannup, with cassiterite in any river sand; Greenbushes, with cassiterite in alluvium of all kinds; Gooseberry Hill, black and white sand, stream; Donnelly River, where zircon forms a large proportion and at times the whole of the concentrates from river sand. At Greenbushes zircon sand could be produced, as a by-product of tin sluicing, in commercial amounts. A sample of sand (30 pounds bulk weight) was analyzed, showing 47.63 of ZrO_2 (Zr, 35.2 per cent). The mineral content, as determined by microscopical examination and analysis, was approximately 71 per cent zircon.

Queensland.--A Government report[151] is to the effect that there was a tendency in Queensland during 1928, on the part of some prospectors, to exploit "nonmetallic minerals," particularly silica, diatomite, zircon, clay, and mica; but that so far no commercial deposits of these minerals had been reported, except in the case of clay. (Twenty-four samples of beach and other sands were tested during the year.)

However, zircon was among the minerals that the Beach Sands Mining Co., at Flat Rock Creek, Tugun, hoped to win through treatment of the sands with a concentrator. A Queensland Government publication[152] reported that this company obtained prospecting rights over a large area of beach sands, extending 4½ miles along the beach at Coolangatta and Currimbin, for the purpose of mining for gold, silver, platinum, and tin, in association with monazite, zircon, tourmaline, topaz, titanium, and other minerals. The company later obtained a mineral lease over 7 acres at Flat Rock Creek, where a treatment station has been erected and where experimental operations have been begun.

150/ Simpson, Edward S., The Rare Metals and Their Distribution in Western Australia; Zirconium: Nat. History and Sci. Soc. of Western Australia Jour., vol. 4, Perth, 1912, pp. 96-98.
151/ Dunstan, B., Chief Government Geologist, Annual Report of the Queensland Geological Survey for the Year 1928: Ann. Rept. of the Under-secretary for Mines, 1928, Brisbane, 1929, p. 127.
152/ Queensland Government Mining Journal, Beach Mining at Tugun: Vol. 29, No. 342, Brisbane, Nov., 1928, p. 469.

An article[153] concerning the Eungella Goldfield (extending from the head of Broken River, along the slopes of the Clarke Range, which lies 40 miles inland from Mackay) is to the effect that the occurrence of zircon in Broken River wash has long been known and that a definite deposit of zircon wash has been reported. The stones are in grains that average a carat in weight, and most of them are rounded in form. Local miners stated that the stones are found in large numbers in Bull Creek, just above an old reef working and in the main river bed 5 miles above it. Zircons occur in some quantity also on Moonlight Creek (Mount Britten) and at the head of Constant Creek. Evidence is strong that the mineral was derived from the disintegration of the trachyte flows that cap the coast range above each of the places mentioned.

New Zealand

In a bulletin of the New Zealand Geological Survey,[154] pannings from Seven-mile, Greymouth, are said to yield the highest percentage of zirconia in New Zealand. Though zircon is not known to occur anywhere in the island in commercial quantities, the best prospects obtained were in pannings from the Whitcombe River, a little above Cataract Creek, near Price Flat, where the zircon is entangled in moss with fine gold. The gem grade of zircon has not been found; however, crystals from Timbrell's Gully are larger than the ordinary ones.

EUROPE

Austria

Venable reports zircons in the Tyrol (Republic of Austria).[155] Garcés[156] reports that H. Whichman[157] in his work concerning the sands of the Tauern en Gschlöss, Tirol (Tyrol), describes zircon, together with epidote, rutile, tourmaline, and apatite. From the minerals encountered, the author concludes that the sands proceed from gneiss and various species of micaceous slate. The presence of zircon and its forms presupposes granite, chloritic slate, and chloritoids (of which no record exists).

153/ Ball, Lionel C., Eungella Goldfield: Queensland Geol. Survey Pub. 229, Dept. of Mines, Brisbane, 1910, p. 33.

154/ Geological Survey Branch, New Zealand, Minerals and Mineral Substances of New Zealand: Geol. Survey Branch Bull. 32 (New Ser.), New Zealand Dept. Sci. and Ind. Res., Wellington, 1927, pp. 107-108.

155/ Venable, Francis P., Work cited, p. 18.

156/ Garcés, V. Soriano, Work cited, 76 pp.

157/ Whichman, H., _____: Min. Mitth., vol. 7, 1886, p. 452.

Czechoslovakia

Venable reports zircon in Bohemia,[158] a Province of the Republic of Czechoslovakia. Pratt reports hainite imbedded in phonolite in northern Bohemia.[159]

France

Frossard reports[160] zircon in the granulite at Pic du Midi, vallée de Lesponne, in the neighborhood of Bagnères-de-Bigorre, Department of Hautes-Pyrénées, France.

According to Garcés,[161] the sands of Mesvrin, in Autun, yield zircon, with a number of other minerals, zircon being second in importance. The crystals measure 0.32 by 0.009 millimeter; they are elongated; and they have faces (110), (112), and at times (001), a form very rare in this mineral. The crystals are light-brown, turning green by reflection. Their specific gravity is 4.5. The accompanying minerals are olivine, rose granite, sphene, chromite, tourmaline, and sapphire. The origin of these zircons seems to be the amphibolites of Marmagne.

M. L. Berthois,[162] in an article upon the heavy minerals of the eruptive and schistose crystalline rocks of Brittany, discusses the form of zircon crystals in these rocks in great detail.

Germany

According to Venable,[163] microscopic crystals of zircon are found in the variegated sandstones in the Black Forest (States of Baden and Württemberg), and zircon is found in the sands in the valley of the Main River (States of Bavaria and Hesse).

Bohm reports[164] zircon at Oberlausitz, Province of Saxony, Prussia.

Brauns,[165] in an article upon the effect of radium rays upon

158/ Venable, Francis P., Work cited, p. 18.
159/ Pratt, Joseph Hyde, Work cited, pp. 11-13.
160/ Frossard, M. Ch.-L., Mineraux des environs de Bagneres-de-Bigorre: Soc. Francaise Mineral., Bull. 8, Paris, 1887, pp. 313-314.
161/ Garcés, V. Soriano, Work cited, p. 13.
162/ Berthois, M. L., Mineraux lourds des roches éruptives et cristallophylliennes de Bretagne: Compt. rend., vol. 188, No. 23, Paris, June 3, 1929, pp. 1506-1508.
163/ Venable, Francis P., Work cited, p. 18.
164/ Bohm, C. Richard, Work cited, pp. 108 and 116.
165/ Brauns, R., Der Einfluss von Radiumstrahlen auf die Farbung von Sanidin, Zirkon and Quarz; Kristallform des Zirkons aus Sanidinit vom Laacher See: Centralb. Mineral., Geol. and Palaon., No. 23, Stuttgart, Dec. 1, 1909, pp. 722-726.

the color of zircon (as well as of sanidine and quartz), refers to zircon from Laacher See, from Niedermendig, and from Pfitsch.

Of the zircon from Laacher See, the author says that in the sanidinite microscopic zircon occurs in two ways for the most part, in some places growing up in geodes (or vugs) and in some places being embedded in sanidinite. It is less frequently found embedded in sanidinite. The first-mentioned form builds small, thin, dominant, prismatically formed crystals. As a rule, the thickness of the prism is less than a millimeter, and the length is from 2 to 4 millimeters.

Specimens from the basaltic lava of Niedermendig the author describes as an embedded, red, long, prismatic zircon crystal, the measurements of which are not reported, as the surfaces are rounded. The only description of the specimens from Pfitsch is that they are colorless crystals, which did not acquire a trace of color when subjected to the radium rays.

Great Britain

A detailed description of the purple zircon in British sedimentary rocks is given by Boswell,[166] from whose work the following excerpts are taken.

The purple or rose-colored variety of zircon is especially characteristic of those British sediments in which minerals derived from regionally metamorphosed rocks are abundant and of fresh appearance. They provide additional evidence that the deposits were formed by denudation of island masses composed of crystalline metamorphic rocks.

British deposits in which purple zircon may be considered an abundant and characteristic constituent of the sediments and in which it is accompanied by other significant detrital minerals are the various divisions of the Carboniferous, Permian, and Bunter rocks, and the Lower Greensand.

In the Carboniferous rocks of Scotland and northern Ireland, purple zircons are found in great abundance, and they are often of a typical deep color and of ovoid form.

In the Permian rocks, to the north of England, purple zircons are by no means uncommon.

The purple zircon is a characteristic and common constituent of Triassic sediments; it appears to be abundant in the Bunter rocks; and it is found throughout Yorkshire, Derbyshire, and Nottinghamshire, the

166/ Boswell, P. G. H., On the Distribution of Purple Zircon in British Sedimentary Rocks: Mineral. Mag. and Jour., vol. 21, Mineral. Soc., London, 1928, pp. 310-317.

Midlands, the Welsh borders, Lancashire, Cheshire, the Vale of Clwyd, Cumberland, Antrim, and Arran.

The localities where purple zircon is plentiful in the Lower Greensand are almost too numerous to mention, but they range throughout the outcrop north and south of the Weald, the Isle of Wight, Berkshire, Oxfordshire, Bedfordshire, Buckinghamshire, Cambridgeshire, and Norfolk.

Greenland

Venable reports zircon in Greenland.167/ According to a report of the U. S. Geological Survey,168/ elpidite is found at one locality in southern Greenland.

Gordon, in his description of the noted cryolite mine at Ivigtut, Greenland, states that eudyalite is of common occurrence in the alkaline syenites about the Tunugdliarfik and Kangerdluarsuk fiords of the Julianehaab district; and that 55 tons were collected for technical purposes, according to K. J. V. Steenstrup. It occurs disseminated in the rock, as well as in veins, and Gordon remarks, perhaps prophetically, that "this district should become an important producer of this mineral if a use for it is ever discovered. . . ."169/

Holland

In the sand dunes of Holland a small percentage of zircon has been found, the zircon and rutile together comprising 0.05 per cent of the mineral constituents of the sand. The zircon has a specific gravity of 4.8 to 4.5.170/

Hungary

An examination of the sands taken from the bed of the Danube River at Budapest revealed that the sands contain opal, granite, magnetite, quartz, calcite, apatite, tourmaline, zircon, rutile, etc. The zircon is among the minerals mentioned as having their origin in the Alps. Zircon occurs in rounded grains and in less rounded forms, colorless prismatic crystals, measuring 0.18 to 0.25 millimeters.171/

167/ Venable, Francis P., Work cited, p. 18.
168/ Pratt, Joseph Hyde, Work cited, pp. 11-13.
169/ Lee, O. Ivan, Work cited, pp. 29-35.
170/ Garcés, V. Soriano, Work cited, p. 17.
171/ Zeitschrift für Krystallographie und Mineralogie, Abstracts, Daten zur mineralogischen Kenntnis des Sandes der Donau: Vol. 53, Leipzig and Berlin, 1914, pp. 61 and 64.

Italy

Baddeleyite is found in Italy, according to Schaller.[172]

Dr. Garcés[173] says that Artini, in his work[174] on the mineralogy of the sands of Tesino, at the extreme southern part of Lake Como, reports the separation of zircon, with gold, pyrite, ilmenite, magnetite, rutile, and other minerals.

Garcés[175] reports also that Uzielli[176] studied various specimens of sand from Porto d'Anzio and from the coast between Naples and Civitavecchia, where he found zircon always accompanied by augite, olivine, monoclinic feldspar, titaniferous magnetite, and calcite. In the sands occurred also quartz, plagioclase, leucite, and apatite. Judging from the distribution of zircon from the mouth of the Volturno River toward the north along the entire coast, from the state of preservation of the crystals, from their prismatic form, from their yellow color, and from the difference of this zircon from that of Vesuvius (which is octohedral and blue), Uzielli inferred that the deposits were chiefly alluvial, except toward the south, where they were probably ocean deposits.

Of the altered zircons, cyrtolite is known in Italy.[177]

Norway

Venable reports zircon in Norway.[178] According to Iddings,[179] zircon occurs very abundantly in certain syenites of Norway. Pratt[180] locates the following noncommercial zirconium-bearing minerals, in southern Norway: rosenbuschite, associated with zircon, elaeolite (eleolite), etc., in syenite; lovenite, associated with elaeolite syenite; eudyalite in syenite; catapleite, hiortdahlite, polymignite, and several other minor zirconium minerals, associated with zircon in syenite. Wöhlerite occurs in zircon syenite on several islands along the southern coast of Norway.

172/ Schaller, Waldemar F., Zirconium and Rare-Earth Minerals: Min. Res. of the United States, 1916, pt. 2, U. S. Geol. Survey, 1917, pp. 377-386.
173/ Garcés, V. Soriano, Work cited, p. 16.
174/ Artini, E., _______: Giorn. min. i petr., vol. 2, 1891, pp. 1-19. (Neues Jahrb., vol. 1, 1892, p. 515.)
175/ Garcés, V. Soriano, Work cited, pp. 10-11.
176/ Uzielli, G., _______: Atti R. Accad. Lincei, vol. 3, 1876. (Neues Jahrb., 1877, p. 303.)
177/ Lee, O. Ivan, Work cited, pp. 29-35.
178/ Venable, Francis P., Work cited, p. 18.
179/ Iddings, Joseph Paxson, Rock Minerals, Their Chemical and Physical Characters and Their Determination in Thin Sections: New York, 1906, 548 pp.
180/ Pratt, Joseph Hyde, Work cited, pp. 11-13.

Bohm[181] reports eudyalite at Brevik (Brevig) and Skudesundskjar, near Barkevik, Norway.

Portugal

A zircon crystal (red hyacinth) was reported at Mount Suimo (Bellas, near Lisbon).[182]

Roumania

Venable reports zircon in Transylvania,[183] a division of the new kingdom of Roumania.

Russia

Since 1920 the Mineralogical Museum of the Academy of Sciences of the Union of the Socialist Soviet Republics and the Research Institute of the North together have been investigating alkaline rocks in the central part of the Kola Peninsula, lying between 67°35' and 67° 55' north latitude, in districts known as Hibina- (or Umptek) and Lujavr-Toundra (or Lujavr-urt). The total area of the massives is 1,600 square kilometers. The country is barren, badly dissected, 1,200 meters above the level of the sea, covered with swamps and forests, and free from snow only two or three months of the year, during the polar summer. However, this region has first rank among mineral districts for the beauty of the specimens, for the pecularity of the minerals, the number of rare combinations, and for the interesting genetic relationships. Fersman[184] says that 15 of the 90 minerals found in the nepheline syenites and their endocontact zones are zircono-titano-silicates (of the eudyalite group): aenigmatite, astrophyllite, eucolite, eudyalite, lavenite, lamprophyllite, lovchorrite, mangan-neptunite, mesodialyte, murmanite, ramsayite, rinkolite, rosenbuschite, titano-elpidite, and wöhlerite.

Kostilewa[185] has given a full description of the work in the Kola Peninsula with respect to zirconium-bearing ores and also of the zircon region of the Urals, a translation of which follows, as well as a translation of his discussion of the economic status of zircon in Russia.

181/ Bohm, C. Richard, Work cited, pp. 108 and 116.
182/ Bello, A., Portugiesische Mineralien; Zircon: Ztschr. Krystall. und Mineral., vol. 53, No. 1, Leipzig and Berlin, 1913, p. 56.
183/ Venable, Francis P., Work cited, p. 18.
184/ Fersman, A. E., Minerals of the Kola Peninsula: Am. Mineral., vol. 11, Mineral. Soc. Am., Nov., 1926, pp. 289-299.
185/ Kostilewa, von E., Zirkonium: Ztschr. prakt. Geol., Mar. 27, 1929, pp. 42-45.

Kola Peninsula

Deposits of eudyalite and eucolite occur in the Chibina Tundra (Umptek) and in the Lujavrut of the Kola Peninsula, where these minerals accompany veinlike segregations in the nepheline syenite. The width of these vein segregations is never great. On an average, their width is $\frac{1}{2}$ meter, and their length is 3 to 5 meters. Their form is generally lenticular. The vein segregations lie along the slopes and upon the plateau-like mountain crest, 300 to 1,000 meters high. The amount of eudyalite in some of these veins reaches 30 to 40 per cent of the weight.

The minerals that most generally accompany the eudyalite are aenigmatite, aegirine, lamprophyres, rinkite ("rinkolite"?), nepheline, feldspar, less often titanium, etc.

Considering the narrow width of the pegmatite veins that contain eudyalite, only a small number of the districts that are rich in such veins can have practical significance.

To such districts belong, as a result of the expedition of A. Fersman (member of the Academy) into the Umptek in 1920, the following:

1. Eudyalite-aenigmatite segregations of the plateau of the South Tschasnatschorrs, which are situated in the valley of the Lutnermajok River, at a distance of 4 to 5 hours from the station of Chibina.--The deposits are on the plateau-like crest of the massif at a height of 1,000 meters above the Imandra Lake (the ascension requires $1\frac{1}{2}$ to 2 hours). They are composed of alluvial beds of primary deposits and of lenticular segregations at Chibinit. A small amount of ore can be collected by hand; a greater yield will require a different method of mining.

2. The northern Zirkus of the Tschasnatschorr, which opens into the Tschasnaiokal, 7 hours from the station of Imandra.--This deposit presents detached accumulations of large bowlders of nepheline syenite strongly enriched with eudyalite at the bottom or base of the Talzirkus (200 to 300 meters high).

In order to calculate the reserves of eudyalite in this district, special research will be necessary; nevertheless temporary observations during the course of the expedition have already indicated that this deposit is richer than that of the southern Tschasnatschorr. For the purposes of production, the great fault blocks will have to be blasted or broken up in some other way.

3. <u>The district of the northern Ljavotschorr, the southern slope of the Ljavajoktals</u>.--This district lies at a distance of 8 to 9 hours, over bad roads, from the station of Imandra. The district is the richest in eudyalite. It is composed of numerous lenticular segregations of eudyalite, lamprophyres, aegimatite, and rinkite (rinkolite?), stretching from northwest to southeast, one of which was opened up by blasting, by the expedition of the Academy of Science and of the Institute of the North. This deposit afforded approximately 160 kilograms of eudyalite, with rinkolite and lamprophyres (a rich material for scientific and museum purposes). These segregations, however, can not yet be worked and assembled; they lie as rich material upon the place of their occurrence. The presence of rinkolite (which is rich in cerium oxide and which can be won together with the eudyalite) increases the value of this deposit.

The eudyalite found in this vein has the following analysis:

	Per cent
SiO_2	49.43
ZrO_2	15.30
TiO_2	1.59
Fe_2O_3	--
Al_2O_3	--
$(Th)_2O_3$	0.52
FeO	5.01
MnO	0.30
CaO	12.29
MgO	--
Na_2O	15.30
K_2O	--
H_2O	1.05
Cl	1.06
Total	100.08

In order to calculate the total amount of eudyalite in this region, a special investigation will be necessary; however, one can now take for granted that here at least 2 tons of eudyalite are at hand. For production, blasting will be necessary.

4. <u>The Privince of Zirkus Sengis in Lujavrut</u>.--Upon the northwest slope of the Sengistschorr, about 400 meters high, are found outcrops of eudyalite-lujavrit, which contains 20 to 40 per cent of eudyalite. The outcroppings of this rock, several meters in width, can be traced many thousand meters along the whole slope of Talzirkus; and they indicate significant accumulations. The eudyalite in this rock has the form of little, pillar-shaped crystals, which are regularly scattered in the mass of feldspar, of nepheline, and of the other minerals of the rock. No special research into the character of these stratifications and no estimate of the quantity of the eudyalite have been attempted. At the present time, it is impossible to reach these deposits, since all connection with the railroad, which is more than 80 kilometers distant, is lacking.

Doubtless new deposits will be found through further searching in Umptek and Lujavrut, which for the practical exploitation of eudyalite will be of importance.

Urals

The most important district for the production of zircon in the Urals is that of the Ilmen Mountains, where the outcropping of zircon is associated with pegmatite veins.

The Ilmen range of mountains is, in a petrographical sense, built up of granite gneiss, nepheline-syenite (miascite), and syenite. The lateral extensions of the granite gneiss are the most significant. The zircon-bearing pegmatite veins lie chiefly in syenite, less often in granite gneiss, and still less often in miascite. The chief distributional area of zircon-bearing veins is the southern part of the main massifs of the Ilmen Mountains, between Lake Miass in the north and Lake Ilmen in the south. It consists of miascite and is inclosed by small outcroppings of syenite rock. Numerous mining works, which have been in existence since 1828, are developing pegmatite veins of almost the same structure. The rock of the veins consists in all the mines almost entirely of white and red potash feldspar, with biotite and a little eleolite. The zircon is in large and small crystals, 2 to 3 centimeters in diameter, inclosed especially in feldspar, less frequently in biotite, and still less frequently in eleolite.

Accessory minerals, which accompany the zircon in the veins, are magnetite, pyrochlore, apatite, etc.

The principal mines that are worthy of note are in the following districts:

<u>On the left bank of the Tscheremschanka River</u>.--A group of deposits, established by Barbot de Marny and F. Blum, lies on the left bank of the Tscheremschanka River. The zircons occur in rather large quantities; they are transparent, brownish-red, but not very large. The largest mine reaches the length of 24 meters by a depth and width of 6 meters. In the year 1837 the largest zircon crystal that was ever found in the Ilmen Mountains was won here. It weighed 3.5 kilograms.

<u>Vicinity of Roschkow Springs</u>.--The mines in the vicinity of Raschkow are the mines of Redikortzew. On the left bank in granite gneiss are two mines, the larger of which is 4 meters long and about 2 meters wide and deep. The zircons found here are pure but not large.

<u>Mines along the Kamenka and Uzkow Springs</u>.--The deepest mine, established in the presence of the acadamician N. Kokscharow, is about 50 meters long, 3 meters deep, and about $1\frac{1}{2}$ meters wide. This mine has yielded numerous and very large crystals, 7 to 8 centimeters long.

<u>The zircon mines of Gasberg</u>.--The largest of these mines, about 8 meters long and approximately $1\frac{1}{2}$ meters deep and wide, has yielded numerous, though not very large, crystals.

<u>The Blumsche mine</u>.--The Blumsche mine, one of the richest, is found in Absturz, on the left bank of Uzkow Springs. It has for some time yielded a quantity of crystals. In the summer of 1926, through the South-Ural-Trust, 480 kilograms of zircon were washed from the deposits of this mine.

It is impossible to calculate the reserves of zircon in the mines described above; for the pegmatite veins have seldom come to the surface, and the whole district is poorly explored. Doubtless in the near future new zircon-bearing veins will be uncovered. The exploitation of zircon by blasting will be certainly very costly and not sufficiently productive, as the occurrenco of zircon in the mines is very scattered and seems to be by chance. Of greater importance are the natural outcrops of pegmatite and the dumps of old mines that accumulated during the working of the mines in the Ilmen Mountains. Only through the washing of these deposits can the extraction of zircon be productive and profitable--a fact that the output of the summer of 1926 has already shown. One may conjecture that through the washing of these deposits in the Ilmen Mountains not less than 1,600 kilograms of zircon can be won.

Besides the deposits in the Ilmen Mountains, in the recent period of the activity of the "Rare Elements" trust, new deposits of zircon sands in the Province of Transbaikalia (in Siberia) have been found. They are in the vicinity of the wolframite occurrences, and their sands contain about 2 per cent of zircon.

Output

The production of zircon in the Ilmen Mountains has already (since 1828 to the present time) been undertaken by Barbot de Marny, Kokscharow, and Blum, exclusively as museum material. First in the eighties, when the foreign market showed interest in zirconium, and when inquiries from abroad concerning zirconium were received, the winning of zircon in the Illmen Mountains, through the washing of mine deads, began. The peak of the industry was in 1887. In the summer of that year 410 to 640 kilograms of zircon arrived at the Petersburg mineral market for further sale abroad. At the same time, however, the foreign inquiries ceased, and the Russian zircon was taken out of trade (commerce), since cheap American zircon had conquered the world market. An enormous slump in prices was the result. From that year on the production of zircons in the Urals ceased or was carried on only by collectors for museum purposes.

In time, interest in the production of zircon revived. From 1925 on, the South-Ural-Trust carried on at the same time its searching for and the production of zircon.

The production of eudyalite has not again been started, unless one takes into consideration the research expedition, under the name of the Mineralogical Museum of the Academy of Science, in the summer of 1925. It was organized in the plateau district of the Southern Tschasnatschorr. It obtained about 20 kilograms of ore, which was rich in eudyalite.

Spain

The zirconiferous sand of Vigo,[186] a port on the northwestern coast of Spain, is of gem grade ("gemmiferous"), like that of Ceylon, and is extraordinarily rich in zircon. The proportions of the different minerals in the sands are shown in the following table.

186/ Garces, V. Soriano, Work cited, 76 pp.

	Per cent
Zircon	67.3
Rutile	7.6
Magnetite	7.1
Quartz	6.7
Feldspar	3.2
Estaurolite	2.9
Cassiterite	1.2
Voigtite ("vigoita")..	1.0
Tourmaline	.3
Limonite	.2
Almandine (garnet) ...	.2
Others	2.2
	99.9

This proportion of 67.3 per cent of zircon is unusually high. Dr. Garcés classifies the crystals observed at Vigo under four different types. The most abundant of these types is a crystal described as a combination of faces that measure (100), narrow, (110), and (111), with slight altitude. The general color of these crystals is that of honey, slightly reddish in tinge, but some crystals are whitish. The two colors give to the sand its characteristic light-brown cast. The Vigo zircons, probably present originally in granites and granite porphyry, are as a rule little worn; they were accumulated in a zone where surge and tide are slight.

Dr. R. de Buen[187] determined the presence of zircon, with quartz, orthoclase, plagioclase, and numerous other minerals, in the bay of Palma de Mallorca, most of the rare and heavy minerals being in the interior part of the bay, where it felt the influence of the port-side river detritus.

Zircon is reported to be in the rich mercury ore of Almaden, Spain, the description by Beck being as follows: "Cinnabar between grains of quartzite and formed by replacement in quartz, also showing sericite, pyrite, and zircon."[188]

Sweden

Zircon on the Island of Alnö, on the southeastern coast of Sweden, when analyzed by P. J. Holmquist, showed the following proportion of mineral oxides:[189]

187/ Buen, R. de, Estudio batilitológico de la bahía de Palma de Mallorca, Trabajos de Oceanografía y Biología marina, dirigidos por el Prof. Dr. Odón de Buen, Director del Institute Español de Oceanografía, Madrid, 1916.

188/ Lindgren, Waldemar, Work cited, p. 495.

189/ Högbom, A. G. (In Upsala, earlier in Stockholm), Mineralien von Alnö; Über das Nephelins-yenitgebiet auf der Insel Alnö: Geol. Fören. Förh., vol. 17, 1895, p. 100.

	Per cent
Zirconium oxide ...	64.94
Silicon oxide	29.68
Titanium oxide	Trace
Manganese oxide ...	.28
Iron oxide	1.15
Hydrogen oxide	3.86
	99.91

Baddeleyite[190/] also has been identified in Sweden, as well as cyrtolite.[191/]

Switzerland

O. Meyer, in 1878, described microscopic twin crystals growing out of the hornblende and the mica and limestone slate of the St. Gothard tunnel. E. Hussak found in the same rocks polysynthetic twinning, which, according to a later notice, Meyer also observed.[192/]

190/ Schaller, Waldemar F., Work cited, pp. 377-386.
191/ Lee, O. Ivan, Work cited, pp. 29-35.
192/ Jahresbericht über die Fortschritte der Chemie für 1878, Mineralogie; Oxyde, Hydroxyde, Oxydhydrate; Zirkon, Rutile, Brookite: Vol. 31, Giessen (Hesse), 1880, p. 1214.

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Scott Turner, Director

INFORMATION CIRCULAR

HAFNIUM

BY

Paul M. Tyler

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

HAFNIUM[1]

By Paul M. Tyler[2]

CONTENTS

INTRODUCTION

Hafnium appears to be among the more abundant of the newly discovered elements. Though quite widely distributed in nature, its compounds chemically so resembled those of zirconium, its sister element, that it escaped detection until 1923, when Coster and Hevesy announced the discovery of the element, to which they gave the name of "hafnium", from Hafniae, the Latin name for Copenhagen, where their research work was performed. Hafnium has so far been found only as a minor constituent of zirconium minerals. Zirconium as an element was discovered more than 140 years ago, but only a few unimportant uses were found for the metal or its compounds until some 18 years ago; it is still classed among the rarer commercial metals. It is not surprising, therefore, that hafnium has not yet found a definite place for itself in industry. Nevertheless it has commanded a considerable interest among scientists; and in 1930 some 70 grams of perfectly pure oxide of hafnium were prepared in Europe. A commercial future for hafnium is already glimpsed in the radio industry, and its high melting point and electronic emissivity have already led to the taking out of

1 The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6457."

2 Chief engineer, rare metals and nonmetals division, U. S. Bureau of Mines assisted by Miss E. P. Youngman of the Bureau of Mines.

patents for its use in radio tubes and incandescent electric lamp filaments and for the cathode surfaces of devices such as X-ray tubes and rectifiers.[3]

Due to the fact that the separation of hafnium (compounds) from zirconium (compounds) is laborious and because of the lack of any extensive demand, hafnium compounds are expensive and not particularly easy to obtain in the market. In the United States practically none of the metal seems to be available, but hafnia (the oxide) and hafnium chloride can be purchased in small quantities at a price of approximately $25 a gram. Ores containing up to about 3 per cent of hafnia can be bought for about 50 cents a pound and ore containing 5.5 per cent may be had for $1.50 a pound.

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In preparing the present paper, the author has depended largely upon the ratherm meager literature, references to which are given whenever possible by means of footnotes. For careful review and constructive criticism of the manuscript, and for additional information some of which has not previously been published, the author is indebted to Mr. O. Ivan Lee.

DESCRIPTION AND PROPERTIES

afnium is element number 72 and falls in the fourth column of the periodic system between zirconium and thorium. The atomic weight as determined by one of the discoverers is probably between 178.64 and 178.59.[4] The metal has the same crystal structure as zirconium (hexagonal) but is roughly twice as heavy. The density, according to the latest determinations by de Boer and Fast is 13.31, about the same as quicksilver and substantially heavier than lead. The atomic volume figures out to 13.42. The melting point is quite high, 2500 degrees on the Kelvin scale (about 2227° C.). According to S. Meyer,[5] hafnium is diamagnetic. The electrical resistance at 0° C. is variously given as 30×10^{-6} ohms or as 41×10^{-6} ohms, and the temperature coefficient of resistance is 0.0044 (0 – 100°).

The chemical properties of hafnium lie between those of zirconium and thorium but are much closer to, and in fact almost identical with, those of zirconium. The high melting point and light emissivity of zirconia are shared by hafnia – hafnium dioxide, HfO_2 – which may be prepared by igniting the sulphate at 1000°. The molecular volume of hafnia is 21.76, that of zirconia 21.50; and the specific gravities are 9.68 and 5.73, respectively.[6] In general any chemical reaction characteristic of zirconium is also characteristic of hafnium, although certain minor differences have been noted by Hevesy, as follows:[7]

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3 Holst, Gilles, and Oosterhuis, Ekko, Electric Discharge Tubes with Cathode Surface of Hafnium: U. S. Patent 1701849 (to Naamlooze Vennootschap Philips' Gloeilampenfabrieken), Feb. 12, 1929: Chem. Abs., vol. 23, No. 6, Am. Chem. Soc., Mar. 20, 1929, p. 1315.

Siemens & Halske A.-G., Incandescence Cathode: Australian Patent 109083, Oct. 14, 1925; German Patent, Dec. 17, 1924; Chem. and Ind., vol. 49, No. 12, London, Mar. 21, 1930, p. 246 of Chem. Abs. sec.

Patent-Treuhand-Ges. f r Elektrische Gl hlampen, Filaments of Hafnium Carbide for Electric Lamps: British Patent 312273 (to General Electric Co., Ltd.), May 23, 1928; Chem. Abs., vol. 24, No. 4, Am. Chem. Soc., Feb. 20, 1930, p. 795.

4 Hevesy, G., The Discovery and Properties of Hafnium: Chem. Rev., vol. 2, No. 1, Am. Chem. Soc., Apr., 1925, p. 18.

5 Mellor, J. W., A Comprehensive Treatise on Inorganic and Theoretical Chemistry: vol. 7, New York, 1927, p. 170.

6 Mellor, J. W., Work cited, pp. 170-171.

7 Hevesy, G., The Chemistry of Hafnium: Chem. News, vol. 127, Sept. 21, 1923, London, p. 186.

While the fluorides and double fluorides of thorium are practically insoluble, the corresponding zirconium compounds, and still more the hafnium compounds, are fairly soluble in cold, very soluble in hot, water. By this method zirconium can easily be separated from hafnium. The mineral is melted with KFHF, and by crystallizing the potassium double fluorides the hafnium concentrates in the mother liquor.

Hafnium oxalate is soluble in excess of oxalic acid. The oxychloride is less soluble than zirconium oxychloride. Hafnium is more basic than zirconium; accordingly the latter is more easily precipitated by ammonia, sodium thiosulphate, etc.; and while zirconium sulphate begins to decompose above 400°, the temperature at which hafnium sulphate undergoes marked decomposition lies about 100° higher. Thorium phosphate is easily dissolved by strong acids, zirconium phosphate much less, whereas hafnium phosphate is found to be still less soluble.

It will be noted that chemically the principal difference is in basicities, and even this is too slight to afford a means for separating the two elements. Actually the only basis for effecting a separation is essentially physical and not chemical; it depends upon differences in solubilities of certain of the compounds.

OCCURRENCE

On purely theoretical grounds it is reasoned[8] that hafnium is not a very rare element. Elements that like hafnium (72) have even atomic numbers are known to be more common than nearby elements in the periodic system (Mendeleef) that have odd atomic numbers. Thus germanium (32) seems to be more abundant than gallium (31); tin (50) than indium (49); and lead (82) than thallium (81).[9] Moreover, a study of the relative abundance of the better known members of the same family in the lithosphere tends to indicate that hafnium is at least more abundant than thorium. The oxides of silicon, titanium, zirconium, and thorium are found in the earth's crust in the percentages, respectively, of 59.09, 1.05, 0.04, and 0.002.

The only known sources of potential supply of hafnium are the various zirconium minerals, notably zircon. Thorium is the higher homologue of hafnium, and it happens that zircons with relatively large thorium (or uranium) contents have been found in several cases to contain unusually large amounts of hafnium. The degree of radio-activity, in fact, is considered to be an index of the amount of hafnium present in zircon or its alteration minerals; the radioactivity is due to uranium and thorium, and hafnium is isomorphous with these elements. So far, however, no hafnium has been found in any thorium mineral.

The ratio of hafnium to zirconium in different minerals varies between wide limits. In the case of minerals associated with basic igneous rocks the ratio is rarely more than 0.02 and often less. In minerals found in acid igneous rocks, such as granites, the ratio is

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8 Lee O. Ivan, The Mineralogy of Hafnium: Chem. Rev., vol. 5 No. 19, 1928, pp. 19-20.

9 A number of exceptions to this rule should be noted, however. Tin (50), for example, is probably less abundant than antimony (51).

much higher, amounting in the case of cyrtolite (altered zircon) to 0.4 and in the case of thortveitite to 0.5. Zircon, the commonest zirconium mineral, occurs in both basic and acid rocks, but in the former association it apparently contains scarcely more than one-third as much hafnia as it does when found in acid or granitic rocks.

List of zirconium minerals known to contain hafnium[1]
(Specific paired analyses are underscored)

Name	Per cent ZrO_2	Per cent HfO_2
Baddeleyite (distinct crystals)	96.5-97.7-98.9	1.2
Syn. Brazilite (fibrous, botryoidal, columnar)	71-93	
Syn. var. (rock) Jacupirangite (crystallized)		
Zirconia ore (1. Favas (alluvial pebbles)	59-74-92.4	0.5-0.5-0.7
Zirconia ore (2. "Zirkite" (mixture) (Brazilite	71-93	
Zirconia ore (2. "Zirkite" (mixture) (Zircon	67	
Zirconia ore (2. "Zirkite" (mixture) (	80-85	
Zirconia ore (2. "Zirkite" (mixture) (Unnamed Zr silicate ((orvillite?)	75	
Catapleite	31.53	0.3
Elpidite	20.28	0.2
Eudialyte	12.20-14.32	0.1-0.2-0.17
Var. Eucolite	12.21	0.2
Polymignite	29.11	0.6
Pyrochlore	2.90	Trace
Rosenbuschite	19.80	0.3
Thortveitite	2	0.5
Var. "Befanamite"	1.3	1.0
Wöhlerite	15.61	0.5
Zircon	64.23	0.98
Var. Hyacinth	64.83	1.2
Var. Hagatalite[2]	39.5	2.5
Var. Oyomalite[2]	38.4	2.5
Var. Alvite	41.98	4.6
Alt. Cyrtolite	52.4	5.5
Alt. Malacon	53.2-65.18	3.4-2.6
Alt. Naegite	49.8	3.5
Zirkelite	51.89	1.0

1 Lee, O. Ivan, Work cited, p. 22.

2 Kimura, K. Zeit. Phys. Chem. vol. 128, 1927, p. 396.

List of zirconium minerals not reported investigated for hafnium[1]

Name	Per cent of ZrO_2
Astrophyllite	1.21-4.97
Beckelite	2.5
Chalcolamprite	5.71
Endeiolite	3.78
Guarinite) Identical?	19.70
Hiortdahlite)	21.48
Johnstrupite	2.84
Lavenite	28.79-28.90
Leucosphenite	3.5
Loranskite (var. wiikite)	20.00
Lorenzenite (ramsayite)	11.92
Mosandrite	7.43
Nohlite (var. samarskite)	2.96
Oliveiraite (alt. euxenite)	63.36
Orvillite (in zircon in "caldasite")	68.04
Riebeckite (arfvedsonite)	0.75-4.7
Soda-catapleite (var. catapleite)	30.80
Uhligite	21.95
Zircon	
Alt. Auerbachite	38-61.53-69

1 Lee, O. Ivan, Work cited, p. 24.

IDENTIFICATION AND ANALZSIS[10]

Although, for authentic determination of the actual hafnium content of minerals and ores, reliance must for some time to come be placed on experienced chemists, the radioactivity of the mineral might aid in preliminary identification and even in an approximate estimation in the case of the different varieties of zircon and its alteration minerals. As different types of altered zircon that have been found to be richest in the new element are pseudomorphic after zircon, it is not difficult to recognize their square pyramidal form, if specimens of various alterations are studied. If such minerals, upon identification as zircons, exhibit pronounced radioactivity, it may be assumed that they contain appreciable quantities of hafnia.

Any chemical reaction that is characteristic of zirconium can be used to identify hafnium also, as, for example, the precipitation of phosphates insoluble in concentrated mineral acids, the coloring of tumeric paper, and so on. As hafnium is always associated with zirconium in nature, the practical analytical problem in dealing with the new element is to determine the hafnium content of zirconium preparations. When such preparations are quite pure, their hafnium content can be readily determined (1) by density measurements or (2) by analysis of any well-defined compound (e.g., one of the double fluorides, the sulphate, or one of the tetrafluorides). But if impurities are present, the method of quantitative X-ray spectroscopy has to be applied.

10 Lee, O. Ivan, Work cited, pp. 24 and 35.

GEOGRAPHICAL DISTRIBUTION

In general, any discussion of the deposits of zirconium covers deposits of hafnium as well. Reference, therefore, is made to the Bureau of Mines information circular on zirconium deposits. A tabulated summary of hafnium mineral occurrences follows:[11]

The occurrence of hafnium minerals by countries[1]

Locality	Mineral	ZrO_2	HfO_2
Africa:			
Diego Suarez	Zircon		0.8
Madagascar	Zircon		0.9
Madagascar	Grey zircon		0.8
Madagascar, Befanamo	"Befanamite," (thortveitite)	1.3	1.0
Madagascar	Malacon (alt. zircon)	53.2	4.
Asia:			
Ceylon	Beccarite (var. zircon)		2.1
Ceylon	Reddish-brown zircon		2.7
Ceylon	Zirkelite	51.89	1.
India (Travancore?)	Zircon from monazite	64.0	1.2
Japan, Naegi Mino Prov.	Naëgite	48.30	7.
Japan, Naegi Mino Prov.	Naëgite	49.8	3.5
Siam, Province of Chantaboon	Zircon (transparent blue)		4.?
Australia: No minerals have yet been definitely reported from the continent as containing hafnium			
Tasmania	Zircon		(2)
Tasmania	Greyish brown zircon		1.1
Europe:			
Austria, Carinthia	Zircon (transparent white)		4.?
France, Espailly (Le Puy)	Zircon (var. hyacinth; transparent red)	64.83	1.2
France, Espailly (Le Puy)	Zircon (var. hyacinth; transparent red)		0.7
France, Espailly (Le Puy)	Zircon (transparent green)		1.1
Italy, Lonedo	Zircon (yellow)		0.7
Italy, Lonedo	Zircon (red)		0.7
Italy, Vicenza	Zircon (transparent green)		0.8
Italy, Vesuvius	Zircon		0.7
Norway, Barkevik	Eudialyte (var. eucolite)	14.47	0.7
Norway, Barkevik	Eudialyte (var. eucolite)	12.21	0.2
Norway, Barkevik	Wöhlerite	15.61	0.5
Norway, Brevik	Zircon		1.0
Norway, Brevik	Reddish-brown zircon	63.2	1.0
Norway, Frederiksvärn	Polymignite	29.11	0.6

11 Lee, O. Ivan, The Mineralogy of Hafnium: Chem. Rev., vol. 5, No. 19, 1928, pp. 32-34.

The occurrence of hafnium minerals by countries - Continued

Locality	Mineral	ZrO_2	HfO_2
Europe - (Continued:			
Norway, Frederiksvärn	Brown zircon	65.2	1.0
Norway, Gjersted	Alvite (alt. zircon)		9.
Norway, Iveland	Thortveitite	2	0.5
Norway, Hitterö	Malacon (alt. zircon)	65.18	2.6
Norway, Kragerö	Alvite (alt. zircon)	41.98	4.6
Norway, Kragerö	Alvite (alt. zircon)		3.
Norway, Krageró	Alvite (alt. zircon)		8.
Norway, Krageró	Alvite (alt. zircon)		15.
Norway	Catapleite	31.52	0.3
Norway, Langesund	Brown zircon		1.7
Norway, Langesund	Rosenbuschite	19.80	0.3
Norway, Larvik	Greyish brown zircon		6.
Norway, Risör	Alvite (alt. zircon)		10.
Norway	Grey syenite (rock)		3.8
Norway, Unneland	Thortveitite	0.8	1.1
Sweden, Alno	Pyrochlore	2.90	Trace
Russia, Miask	Greyish brown zircon	64.22	1.1
Russia, Rojkow Kliutsch, Ural	Brown zircon		0.5
Russia, Kola	Eudialyte (red)		0.1
North America:			
Canada, Eganville, Ont.	Brown zircon		1.2
Canada, Renfrew County, Ont.	Zircon		0.6
Canada, Henvey Tp., Parry Sound District, Ont.	Cyrtolite, black (alt. zircon)		2.11
Greenland	Zircon		0.8
Greenland	Catapleite	31.53	0.3
Greenland, Narsarsuk	Elpidite	20.28	0.2
Greenland, Narsarsuk	Reddish-brown zircon		0.8
Greenland, Narsarsuk	Eudialyte (red)	12.20	0.2
Greenland, Narsarsuk	Eudialyte (brown)	12-16	0.6
Greenland, Kangerdluarsuk	Eudialyte	14.32	0.17
United States, Connecticut	Zircon		1.0
United States, Rockport, Mass.	Cyrtolite (alt. zircon)	40.	9.
United States, Rockport, Mass.	Cyrtolite (alt. zircon)	44.	17.(?)
United States, Bedford, N. Y.	Cyrtolite (alt. zircon)	52.4	5.5
United States, N. Car.	Grey zircon		4.(?)
United States, N. Car.	Brown zircon		1.3
United States, Henderson Co., N. Car.	Zircon		4.0(?)
Oceania: No minerals have yet been reported definitely as containing hafnium			
South America:			
Brazil	Baddeleyite	97.7	1.2
Brazil	Fava	92.42	0.7
Brazil	Fava (shell)	59.	0.5
Brazil	Fava (nucleus)	74.	0.5
Brazil	Zircon separated from monazite sand	64.	0.4
Brazil, Caldas	Zircon		1.8
Brazil, Minas Geraes	Zircon		1.0

1 Lee, O. Ivan, Work cited, pp. 32-34. 2 "Very considerable."

The richest hafnium ores are the altered zircons (especially cyrtolite). The unaltered zircons, the native oxide (baddeleyite) and eudialyte, being more abundant are the more important of the leaner hafnium ores.

United States.- Unaltered zircon (the only commercial hafnium ore in the United States), which usually contains less than 2 per cent of hafnium oxide, has been mined intermittently in Henderson County, N. C., and at Pablo Beach, Fla., and has been reported in Virginia, near Ashland. It also has been found near Indiahoma, Commanche County, Okla. Baddeleyite has been reported in Montana, and eudialyte has been found in Arkansas.

Of the altered zircons, cyrtolite, which is the most widely distributed, occurs in at least eight States, as follows:

California (Southern).
Colorado, Mount Antero.
Connecticut, Branchville.
Massachusetts, Rockport.
New York, Westchester County, near Bedford.
North Carolina, Mitchell and Henderson Counties (three localities).
Pennsylvania (five localities).
Texas, Llano.

The cyrtolite at Rockport (Mass.) probably contains as much as 17 or even 20 per cent hafnium, but this ore is very rare, as it is in Connecticut and Colorado also. As a 24-hour exposure of the cyrtolite of Texas yielded good radiographs, it is assumed that this ore likewise is high in hafnia, although no analysis has been reported. It is stated that cyrtolite was abundant in the famous Baringer Hill locality when the yttria minerals were being mined there. The radioactive cyrtolite of New York, which is as abundant as that of Texas, is unusually high in hafnia and appears to be the most promising domestic source of supply. A large feldspar mine situated near Bedford in the hills of Westchester County, within about 40 miles of New York City, has yielded a variety of unusual mineral specimens. Among these minerals was cyrtolite, which had been known for some time to contain a small percentage of uranium and probably rare earths. Mr Lee sent a sample of this mineral to Prof. C. James of the University of New Hampshire, who reported the presence of some 10 per cent of hafnia in the mixed oxides (HfO_2 and ZrO_2). James's original analysis was confirmed by Hevesy, who found approximately 9.5 per cent of hafnium oxide in the mixed zirconium oxides, which totalled 52.44 per cent. These unusually high percentages of hafnia appear to be rather consistently maintained in different samples from this locality, although the yttria earths content varies considerably. The deposit is not large, but specimens weighing 10 or 12 pounds are not uncommon; and Professor James actually worked up a fairly large quantity of this material for hafnium and its associated zirconium.[12]

Foreign countries.- Of the ores richest in hafnia, the altered zircon, cyrtolite, occurs in India, Madagascar, Italy, Sweden, and Canada (Hybla and Parry Sound district, Ontario). Alvite, which does not seem to be common but which was useful to early investigators of hafnium, occurs in Norway.

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12 Lee, O. Ivan, Work cited, pp. 25-29.

Of the ores lean in hafnia, the zircon of commerce has come from the monazite sands of Travancore, India, and of Brazil, and from the pegmatites of southern Norway.

Baddeleyite (the native oxide), known for 37 years, occurs in Ceylon, Italy, Sweden, and especially in Brazil, in the States of Minas Geraes and São Paulo. Bowlders in the Brazilian deposit have been found that weighed as much as 30 tons; such bowlders would probably contain 300 to 400 pounds of hafnium oxide.

The complex eudialyte, said to be a third hafnium mineral of great interest, is reported to be very abundant in Hibina-Toundra and Lujavr-urt, in the Kola Peninsula of Russian Lapland. (Although specimens of about a dozen others of the rarer zirconium minerals have been collected from the same region, only the eudialyte has been analyzed for hafnium.) Eudialyte is reported to be of common occurrence in the alkaline syenites in Greenland, about the fiords of Tunugdliarfik and Kangerdluarsuk, in the Julianehaab district.

HISTORY

D. Coster and G. von Hevesy are credited with the discovery of hafnium. They made their announcement on January 2, 1923,[13] after measuring the X-ray spectra of a number of zirconium minerals and finding some characteristic lines in the L-series belonging to no previously known element. The X-ray method was used for the quantitative determination of hafnium by comparing the intensities of the lines of a known amount of the neighboring element tantalum (number 73 in the periodic table) with those of the unknown element.

After the discovery of hafnium there was some controversy following the claim by G. Urbain that the new element was identical with that which he had described as "celtium" in 1911 and which he had found in some residues remaining after the separation of the lutecium-ytterbium fractions of the rare earths. However, the existence of "celtium" as a new element was later disproved.[14] Dr. Alexander Scott, likewise, was considered as a possible olaimant of the honor of discovering the new element. In 1913 he stated that in analyzing samples of a black sand from New Zealand he had extracted a cream-colored sand containing about 75 per cent titanium oxide and also a highly refractory residue which he considered a new oxide. Coster and Hevesy, however, after examining some of the material sent them by Doctor Scott informed him that it failed to yield any hafnium lines in the spectrum. A complete discussion of their findings, showing that the so-called "element celtium" was widely different from hafnium, particularly in its chemical reactions, was made by Coster and Hevesy[15] soon after the original announcement of their own discovery; and two years later Hevesy[16] summarized the various attempts to find new elements in zirconium minerals since zirconium itself was identified by Klaproth in 1789, as follows:

> In 1845 Svanberg expressed the view that zirconium minerals contained an element similar to zirconium, which he named norium. Sjögren, another Swedish chemist, a few years later, again thought he had found Svanberg's norium in the zirconium mineral catapleiite. Then came Nylander's announcement (1869) of the discovery of jargonium, an

13 Coster, D., and Hevesy, G., On the Missing Element of Atomic Number 72: Nature, vol. 111, London, Jan. 20, 1923, p. 79.

14 Mellor, J. W., Work cited, p. 166.

15 Coster, D., and Hevesy, G., On the New Element Hafnium: Nature, vol. 111, London, Feb. 24, 1923, p. 252.

16 Hevesy, G., The Discovery and Properties of Hafnium: Chem. Rev., vol. 2, No. 1, April, 1925, pp. 10-11.

element similar to zirconium, but having a lower atomic weight; and in the same year Church thought he had discovered the new element nigrium. We may add to this list the announcement of Ogawa (nipponium, 1908), and of Hoffmann and Prandtl (1901), who believed they had found a new earth ("euxen earth") in the mineral euxenite. The genuineness of all these announcements was later disproved by different investigators, including Marignac, Weibull, and Hauser, who showed the identity of norium, and so on, with zirconium. Most of these announcements were made on the basis of peculiar chemical reactions believed not to be characteristic of zirconium, and from this fact alone we can straightway conclude that the above announcements have nothing to do with the presence of hafnium, as zirconium and hafnium show the same chemical reactions. Nevertheless, the latter element was without exception undoubtedly present in the investigated samples.

SEPARATION OF ZIRCONIUM AND HAFNIUM

A number of methods that Coster and Hevesy [17] have described for the separation of zirconium and hafnium include fractional crystallization or precipitation of (1) zirconium and hafnium sulphates, (2) fluorides or double fluorides, (3) oxalates in presence of fluorides, (4) hydroxides, (5) oxyhalides, and (6) organic compounds.

The methods of Van Arkel and De Boer[18] for the separation of zirconium and hafnium depend upon (1) fractional precipitation of oxalates from solutions free of fluorides, (2) adding phosphorus pentachloride to mixed hafnium and zirconium halides and fractional distillation, (3) adding zirconium and hafnium mixture to concentrated sulphuric acid and fractional precipitation by means of a base, and (4) by fractional precipitation with hydrogen peroxide and organic compounds.

A recent German patent[19] covers the separation by the fractional crystallization, sublimation, and precipitation of their halogen compounds (except fluorides); special mention is made of the oxychlorides, due to their difference in solubility.

PREPARATION OF HAFNIUM AND ZIRCONIUM

A method for the preparation of pure zirconium and hafnium, which is a summary made by Marden[20] of an article by Van Arkel and De Boer,[21] follows:

> Their process consists in heating zirconium or hafnium tetraiodide in a pyrex bulb to 600° C. The bulb contains, in addition to the chemical compounds, a tungsten filament heated to 2,000° C. The

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17 Coster, Dirk, and von Hevesy Georg, Separation of Hafnium and Zirconium: Canadian Patents 275682 to 275687, incl. (to N. V. Philips' Gloeilampenfabricken, Eindhoven, The Netherlands), Nov. 22, 1927.

18 De Boer, Jan Hendrik, Separation of Hafnium and Zirconium: Canadian Patents 275692 and 275695 (to N. V. Philips' Gloeilampenfabricken), Nov. 22, 1927.
Van Arkel, A. E., and De Boer, Jan Hendrik Separation of Hafnium and Zirconium: Canadian Patents 275688, 275691, 275693, and 275694 (to N. V. Philips' Gloeilampenfabricken), Nov. 22, 1927.

19 Ger. Pat. 492754, *vide* Foote-Prints, vol. 3, No. 2, 1930 pp. 334.

20 Marden J. W., Titanium and Zirconium: Min. Ind., 1925, vol. 34, McGraw-Hill Book Co. (Inc.), 1926, p. 697.

21 Van Arkel, ________ and De Boer ________, ________: Ztschr. anorg. Chem., vol. 148, 1925, p. 345.

tetraiodides vaporize at this temperature and dissociate when they come in contact with the tungsten filament at 2,000°, depositing the metal on the filament. It is claimed that it is possible to make filaments having a diameter of 2 to 4 millimeters in thickness and that these metals when made in this way are extremely soft and pliable, resembling copper.

PREPARATION OF PURE HAFNIUM SALTS[22]

A 60 per cent hafnium salt prepared by the hydrofluoric acid-borax process was dissolved in hydrofluoric acid, precipitation was brought about with sodium hydroxide, the precipitate was washed free from fluoride, phosphate and borate, and dissolved in hydrochloric acid. By treatment of this precipitate according to the sulphuric acid method of separation, a 100 per cent hafnium fraction was obtained on the second treatment. Hafnium phosphate may be converted into the hydroxide by treatment with hydrofluoric acid and potassium hydroxide, solution in hydrochloric acid, and precipitation by ammonium hydroxide. Contamination with silicic acid from hydrofluosilicic acid in the hydrofluoric acid may be removed by evaporating the hydrofluoric acid solution to dryness and taking up with hydrochloric acid. After removal of silica the hydrochloric acid solution is evaporated; crystallization and recrystallization give the oxychloride $HfOCl_2.8H_2O$. Washing with alcoholic hydrochloric acid removes iron salts. The hydrochloric acid should not be too concentrated, because of the tendency to form a complex hafnium chloro acid. Ignition yields white oxide of hafnium.

PRODUCTION AND TRADE

The first hafnium salts produced in the United States were made by the late Prof. C. James of the University of New Hampshire in 1924 from cyrtolite from near Bedford, N. Y. However, although the zirconium salt which he prepared concomitantly appears to have been free from hafnium, the hafnium salt was unfortunately contaminated in process by zirconium through the interchange of fractions by careless workmen who had access to the laboratory. To date, therefore, pure hafnium salts have apparently not been prepared in the Western Hemisphere. Apparently the metal likewise has not been produced in this country, but a small sample of the first metallic hafnium ever made was presented by G. Hevesy for permanent exhibition in the element collection being made by Dr. George F. Kunz at the American Museum of Natural History in New York City.

Practically all the hafnium and its compounds that have been isolated so far have been consumed in scientific and experimental work, and production has of course been confined to a small laboratory scale. In 1930, however, de Boer and Fast are said to have prepared some 70 grams of 100 per cent hafnium oxide from which they made some pure, ductile metal, as well as a series of hafnium-zirconium alloys.

Domestic supplies are imported and may be obtained through various dealers in rare metals and chemicals, as for example:

Foote Mineral Co., (Inc.), 1609 Summer St., Philadelphia, Pa.
A. D. Mackay, 26 Cortlandt St., New York, N. Y.

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22 de Boer, J. H., and Broos, J., Ztschr. anorg. allgem. Chem., vol. 187 (1930), p. 190; Foote-Prints, vol. 3 (1930) No. 2, p. 34.

I. C. 6458
July, 1931

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING LAWS OF CUBA[1]

By I. Aitkens[2]

PREFATORY NOTE

This paper presents one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the laws of Cuba, prepared from the best available information in Washington, has been checked against the answers made by the American Ambassador, Harry F. Guggenheim, to a questionnaire of the Bureau of Mines transmitted through the courtesy of the Department of State. The assistance of the Commercial Laws Division of the Bureau of Foreign and Domestic Commerce in assembling material is gratefully acknowledged.

INTRODUCTION

The basic mining law of Cuba is contained in the basic Spanish mining decree having the effect of law December 29, 1868. This legislation has remained substantially unchanged, with the exception of a few minor administrative changes made by the present Government. These changes provide for the register of mines, oil wells, and operations of such nature, and also provide for the keeping of records showing the value of minerals taken out.

By Royal decree of April 17, 1883, the law of mines enacted for the Peninsula (Spain) on July 6, 1859, and the General Bases of December 29, 1868, were declared in force in Cuba. In order to encourage the mining of iron ore, in 1883 the Crown of Spain issued a royal decree to the following effect:[3]

1 The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from the U. S. Bureau of Mines Information Circular 6458."
2 Rare metals and nonmetals division, U. S. Bureau of Mines.
3 Porter, Robert P., Industrial Cuba: New York and London, 1899, pp. 319-20.

That for a period of twenty years the mining companies should be free from all taxes on the surface area of all claims of iron or combustibles; that ores of all classes should be free from all export taxes; that combustibles and iron ore should be exempted from the three per cent tax on raw materials; that mining and metallurgical companies should be free from all other imposts; that for a period of five years the mining companies should be exempt from the payment of duties and all machinery or materials required for working and transporting the ore; that vessels entering in ballast and sailing with ore should pay a duty of five cents per ton navigation dues, and that vessels entering with cargo destined for the mining companies should pay $1.30 per ton navigation and port dues on all such cargo, and on the remainder of the cargo as per general tariff.

Later, by Decree of September 28, 1914, slight changes were made in the mining law. The principal changes made at this time referred to test pits and damages resulting therefrom. On June 12, 1916, the Law of Labor Accidents was passed, and on July 31, 1917, a law was passed placing a 6 per cent additional tax on net income on mining property.

The law of July 1, 1920, imposes a royalty fee of 20 cents per hectare on land to which mining development rights have been granted under concession from the Cuban Government, whether these rights have been availed of in exploitation or not.

The "Gaceta Oficial" of Feb. 9, 1924, published a decree regarding the collection of royalties on mining concessions in Cuba, and on May 15, 1924, a law was passed further regulating State taxes as imposed by law of 1917.

Presidential Decree 596 of April 29, 1928, comprises regulations regarding the Law of Labor Accidents, dated June 12, 1916; and Presidential Decree 768 of June 7, 1930, contains several amendments to the mining law of September 28, 1914, relating to applications for oil concessions.

RIGHTS OF FOREIGNERS

Americans may freely explore and may own and operate mines on the same terms as Cubans. (Art. 10, Decree-Bases of Dec. 29, 1868.) It is not necessary to incorporate under the laws of the country, and any Cuban or foreigner may freely make test pits or excavations not exceeding 10 meters in length or depth for the purpose of discovering minerals.

Article 10 of the Decree Bases of 1868 reads as follows:

> Any Spaniard or foreigner may, without restriction, dig test pits or make excavations on land of public ownership not to exceed 10 meters in length or depth, for the purpose of discovering minerals. Permission shall not be required for this purpose, but the local authority must be informed thereof.

CLASSIFICATION OF MINERAL SUBSTANCES

All useful substances of the mineral kingdom, whether found on the surface or in the subsoil and irrespective of origin or form in which they are found, are divided under the Cuban law into three classes.

Class 1 includes mineral products of an earthy nature, silicious stones, slates, sandstone, granites, basalt, limestones, gypsum, sands, chalks, argillaceous earth, and in general, all building materials which may be quarried.

Class 2 includes placers alluvial deposits, iron ores including bog iron ore, emery, ochers (and "almagras"), slag and mine dumps (the result of prior workings), peat bogs; "pyritous, aluminous, magnesian and Fuller's earth"; saltpeter beds, phosphorites, baryta, fluorspar, steatite, kaolin, and clays.

Class 3 includes deposits of metalliferous substances, anthracite, pit coal, lignite, asphaltum, and mineral tars, petroleum and mineral oils, graphite, saline substances, including alkaline and terreo-alkaline (alkaline earth) salts, whether in a solid state or dissolved in water; copperas, sulphur, and precious stones. Subterranean waters are considered as belonging to this group. (Arts. 1-4, Decree-Bases of Dec. 29, 1868.)

OWNERSHIP

The State is the owner of all minerals found in the subsoil, and no one may dispose of them without a concession from the Government. (Art. 2, Law of July 6, 1859, and Art. 6, Decree-Bases of Dec. 29, 1868.) The ownership of the surface and the subsoil are separate.

However, fee simple ownership is possible. One may be the owner of both the surface and subsoil. If the owner of the surface obtains a concession for the minerals in the subsoil, or if the concessionnaire of the minerals in the subsoil acquires ownership of the surface, he thereby acquires fee simple ownership.

The <u>soil</u> may be private or public property, and the owner never loses his right thereto nor his privilege of utilizing it, except in cases of condemnation. The <u>subsoil</u> is under the dominion of the State, and it may abandon the same to the public use, grant it gratuitously to the owner, or alienate it by means of a surface tax to individuals or companies requesting it. (Art. 6, Decree-Bases of Dec. 29, 1868.)

Article 5 (Bases), defines soil and subsoil as follows:

> The soil includes the surface, properly speaking, and furthermore the depth to which the work of its owner may have attained, whether in cultivation, for building, or the laying of foundations, or for any other purpose other than mining.
>
> The subsoil extends indefinitely in depth from where the soil ceases.

The substances included in Class 1, are of public use when situated on lands of public ownership. When on private lands, the State cedes them to the owner of the surface. (Art. 7, Decree-Bases of Dec. 29, 1868.)

The substances included in Class 2, when on private lands, are reserved by the State to grant them to whomsoever may request their development if the owner should not work them himself; provided that the enterprise is declared of public utility and the owner is indemnified for the surface condemned and damage caused. (Art. 8, Decree-Bases of Dec. 29, 1868.)

The substances mentioned in the third class may be worked only by virtue of a concession granted by the Government; in that case they are considered property separate from the soil. In cases of a declaration of public utility, condemnation, or the like, proper indemnity is made. (Art. 9, Decree-Bases of Dec. 29, 1868.)

PRELIMINARY PROSPECTING

On public lands no permit to explore is required, but notice must be given in advance to the mayor of the locality. Explorations may not be made on private lands without the permission of the owner. (Art. 10, Decree-Bases of Dec. 29, 1868, and Art. 8, Law of July 6, 1859.)

If the owner of the land is opposed to exploration, the prospector may appeal to the Governor of the Province, who can give or refuse the necessary authorization. Recourse may be hand to the President of the Republic against the decision of the Governor. (Arts. 21, 22, 23, 24, and 25 of Law of Sept. 28, 1914.) However, the owner may require that the prospector give bond to pay for damages which he may cause. (Art. 26, of Law of Sept. 28, 1914.)

Article 26 of the Law of September 28, 1914, reads as follows:

> The owners of the land shall always have the right to require that the prospector post a bond to guarantee the payment of any damages that may be caused by the test pits and the miner is further obliged to pay for all damages which he may subsequently cause to the property.

Prohibitions

The law stipulates that no pits shall be dug or other mining workings made at a distance of less than 40 meters from a building, railroad, road, canal, well, water trough, or other public easement. Also that a distance of 1,400 meters from fortifications must be adhered to, unless permission is obtained from the military authorities.

If public services or easements are affected, authority must be granted by the governor; and when public buildings are concerned, permission must be received from the owners. (Art. 12, Law of July 6, 1859.)

Further Exploration

On free lands before undertaking prospecting work more extensive and important than test pits - such as shafts, tunnels, ditches or excavations - an application must be filed in writing with the governor of the province, requesting permission to make explorations. (Art. 21, Law of July 6, 1859.) If the owners of gardens, orchards, and irrigated farms through which it may be necessary to continue the works started, should refuse access, the governor shall grant permission to make these explorations. (Art. 25, Law of July 6, 1859.)

The application for exploration or registry may be filed without the consent of the owner (art. 20, Law of July 6, 1859). However, within a period of 30 days, counted from the presentation of any petition for exploration or registry, the interested persons shall be obliged to show either the consent or refusal of the owner of the land, or to declare in writing the date on which they asked for the authorization (Art. 28, Regulations).

CONCESSIONS

A concession from the Government is necessary to mine minerals in the subsoil, and application for the concession shall be made to the Governor of the Province. (Art. 2, Law of July 6, 1859.) Although the official granting such permit has not wide discretionary powers, he may refuse to grant it. However, the Government may not refuse to grant a concession if there is sufficient free land and if all legal requirements are fulfilled. (Art. 15, Decree-Bases of Dec. 29, 1868.) Proceedings are initiated and conducted before the Provincial Government and the technical part is under the Bureau of Forests and Mines of the Province. The President of the Republic grants the concession. (Art. 2, Law of July 6, 1859.)

Duration of Grant or Concession

Such grants are perpetual. However, a ground rent or surface tax is required. (Art. 19, Decree-Bases of Dec. 29, 1868.)

Units of Measure

The "pertenencia" or unit of measure for mining concessions, with regard to the substances of the second and third classes, is a solid with a square basis of 100 meters, measured horizontally in the direction which may be indicated by the petitioner, and of an indefinite depth. For the first class, the depth stops where the workable material is exhausted. (Art. 11, Decree-Bases of Dec. 29, 1868.)

Size of a Concession

Concessions have no maximum limit of area, but they may not be less than four hectares - 40,000 square meters (Art. 13, Law of July 6, 1859).

Any number of pertenencias over four may be granted to one person, provided the units forming the concession are joined without a break. All of them which together form one concession must be grouped in such manner that their continuity be uninterrupted, so that those contiguous to each other abut for the entire length of any of their sides. (Art. 12, Decree-Bases of Dec. 29, 1868.)

Applications for Mining Concessions

To acquire one or more mining concessions, one of two methods may be employed - namely, exploration or registration. Both in exploration and registration priority of an application confers a preferred right to a concession (Art. 20, Law of July 6, 1859).

Priority of Right to a Concession

The prospector has no prior rights for explorations which he may have made. (Art. 31, Law of Sept. 28, 1914.) However, priority of the request in the presentation of a petition gives a preferred right. If substances of the second class are in question, the owner is given the preferred right, if he binds himself to work his concession within the period which the administration may indicate and which shall not exceed thirty days (Art. 16, Decree-Bases of Dec. 29, 1868.) If the same land contains substances of the second and third classes and it is impossible to work both at the same time, they shall be granted to the first petitioner. If this petitioner asks to exploit the substances of the third class, he may extend his operations over those of the second class as well; but if he desires to exploit the substances of the second class only, when those substances are exhausted, a new concession will be necessary for the exploitation of the third class mineral substances (Art. 20, Decree-Bases of Dec. 29, 1868).

Surveys for Concessions

A petition to the governor shall clearly state all the circumstances of the concession requested, and after the proper proceedings have been instituted as prescribed by the regulations, the governor shall order that the concession be surveyed (Art. 15, Decree-Bases of Dec. 29, 1868).

No survey shall be made unless some mineral substances of the three classes described above have been discovered, and according to the opinion of the engineer, the petitioner is not including property for which a previous consent of the governor should have been obtained. (Art. 29, Law of July 6, 1859.)

If upon examination of the petition it is found that the legal requirements have been performed, the engineer shall immediately survey the claim or claims according to the surface plan.

RIGHTS OF SURFACE OWNERS

When the land is private property, if the owner binds himself to work the land, beginning to do so within the period fixed by the governor (three months), he shall be preferred to strangers. Working of such substances shall not be consented to without the special permission of the owner. The exception being when same is to be used for pottery works, or any branch of the "fabrile" industry. The governors may grant authorization to work private lands for these purposes to any person requesting it, after the institution of proceedings by the governor with a hearing of the owner of the land and after receiving the report of a mining engineer and the provincial council. (Art. 4, Law of July 6, 1859.)

However, such claimant to the mineral rights must adjust his affairs amicably with the owner of the surface if possible. If the owner protests against the digging or boring, the claimant has the right to buy the property, lease it, or work it. He must pay such owner damages, or a price that if not mutually agreed on, is determined by a board of appraisers, as indicated in a brief outline of the mineral wealth of Cuba, compiled by Bureau of Information, Department of Agriculture, Commerce and Labor, Havana, Cuba, in 1921.

Whenever possible, it is customary to arrange with the owner to give a royalty on whatever ore or oil he may remove from below the surface. This arrangement must be acknowledged before a notary and registered in the nearest municipality. In Cuba, such arrangements have proved eminently satisfactory in mining operations and are intended to be fair to all parties concerned.

FORFEITURE

When the owner has omitted to pay the fixed annual tax, his mining concessions shall be subject to forfeiture. If after being ordered to pay by an injunction, he shall not have acquitted the obligations within 15 days, such concession is deemed forfeited. If the owner proves to be insolvent, such concession also becomes forfeited. The pronunciation of forfeiture is mandatory in the Spanish law.

When forfeiture is declared, the mine shall be sold at public auction. The authorities shall keep the amount of the taxes; the amount for expenses incurred in the proceedings; and 5 per cent of the total price obtained by the administrative authorities. The remainder, if any, shall be turned over to the bankrupt mine owner.

After three successive auctions, if no purchaser is found, the land shall be declared free. (Art. 23, Decree-Bases of Dec. 29, 1868.)

The ownership of mining claims, dumps, or scoriae may be lost or forfeited for five reasons, as follows:

1. Nonfulfillment of conditions.
2. Bad management.
3. Insolvency.
4. Abandonment.
5. Renunciation.

Notwithstanding the foregoing provisions, mining companies may suspend their work for a period of two years without forfeiture, provided they justify their action with good reasons. (Art. 65, Law of July 6, 1859.) The governors shall declare the forfeiture after proceedings of investigation have been instituted, either officially or at the instance of another by means of a registry. (Art. 68, Law of July 6, 1859.)

Decisions declaring forfeiture according to article 68 of the Law of July 6, 1859, are excepted. In these cases an administrative appeal lies to the provincial council, with an appeal to the council of state by the former concessionnaire. Both appeals must be filed within 30 days. (Art. 88, Law of July 6, 1859.)

ROYALTIES

The law of July 1, 1920, imposes a royalty fee of 20 cents per hectare on land to which mining development rights have been granted under concession from the Cuban Government, whether these rights have been availed of in exploitation or not.

After this law came into force, no action was taken for collection of said royalties or for the reversion of such concessions to the State in case of nonpayment, as provided for under laws previously in operation. This delay was due to the ambiguity of article 174 of the Basic Regulations for Cuban Mining as of September 28, 1914, which left in doubt what governmental authority should institute the abrogation proceedings in case of defaulted payments.

The "Gaceta Official" of February 9, 1924, published a Presidential Decree which provides that article 174 shall be altered to require that at the end of each fiscal year all the Federal tax-collecting agencies shall report to the Jefaturas de Minas all of the existing mining concessions, and state with necessary details those who fail to pay the prescribed royalties.

The Jefaturas de Minas then reports to the Provincial Government all delinquents, and the governor issues to each delinquent company a notification stating three provisions, as follows:

(a) That annulment of the concession is under consideration.

(b) That demand is made that each concessionaire shall liquidate his past-due royalty fee to the State, and that annulment of concession is pending.

(c) That notification is made to the Administration of Taxes and Imposts to continue pressure upon the delinquents for collection of the amount due to the State under the royalty regulations.

The Provincial Government summons the interested parties, allowing five days for reply to this summons, after which time the governor submits to the respective fiscal zone a copy of the preliminary action for annulment.

Fifteen days after date of notification of the delinquent, if evidence of payment of such royalty has not been presented, the governor files with the Secretary of Agriculture, Commerce and Labor, the documents covering the delinquent cases along with his recommendations, and final action is taken according to the law.

GENERAL TUNNELS

Every owner of a mining claim is obliged to permit the passage of a general gallery (drainage tunnel); and to respect the supports of such gallery, abstaining from extracting minerals within 2 meters of the walls of the tunnel unless he properly strengthens said walls at his own expense.

The price for the services of drainage, ventilation, and extraction furnished by the constructor of a tunnel or gallery to any miner shall be agreed upon by the interested persons, and if an agreement can not be reached, a valuation shall be made by experts named by both parties and another appointed by the governor. The amount shall be decided upon in view of such expert report and after taking into account the circumstances surrounding the case in question. (Art. 44, Law of July 6, 1859.)

EASEMENTS

To drive general galleries for exploration, drainage, or transportation it is necessary to petition for them. If these works are to cross claims already granted, the petitioner must previously come to an agreement with the owners in case of the discovery of minerals. (Art. 18, Decree-Bases of Dec. 29, 1868.)

Also, every miner is compelled to facilitate the ventilation of the adjoining mines. He shall be subject to the easement of the course of waters from adjoining mines toward the general drainage and also to the police regulations. (Art. 24, Decree-Bases of Dec. 29, 1868.)

Miners may obtain the free and full use of all or part of the surface of their claims for warehouses, workshops, buddles, reduction works, slag and waste piles, roads, and other similar purposes, all within the requirements of their industry. If agreement can not be reached with owners, they shall petition the governor of the province for the immediate application of the law of condemnation of property; which is proper in these cases and shall have effect within two months. If roads must be extended or opened beyond the mining claims, they shall be subject to the general provisions governing road. (Art. 56, Law of July 6, 1859.)

PROPOSED NEW LEGISLATION

The National Commission for Economic Defense made a study of the economic resources of the Republic of Cuba and proposed a mining law dealing with mineral combustibles. This study was made prior to October, 1929, but to date no legislation has been passed, and from all reports it does not appear practicable to settle this problem in the near future.

The so-called Vasquez Bello Bill, concerning petroleum and coal concessions, is now pending (January, 1931).

CHANGES IN CUBAN LAW RELATING TO OIL CONCESSIONS

The Official Gazette of June 11, 1930, published a Presidential Decree, No. 768, June 7, 1930, in which several amendments were made to the Mining Law of September 28, 1914. These amendments relate to applications for oil concessions. By this new decree a concessionnaire is granted a year's time to make payment; such payments vary in amount according to the number of hectares claimed. Furthermore all applicants are pledged by the Government to fulfill the following requirements:

> To present a report from a competent engineer or geologist within one year; also to notify the Government of the engineer's name, within six months from the date of presentation of the application.
>
> To deposit within ten days from receipt of notice from the Government, the amount designated for publication of proper notices required by articles 50, 51, and 52 of the regulations.
>
> To pay the Government, in addition to the profits tax, a royalty of 5 per cent free of expense, upon all oil producing wells included in the concession.

This new decree granting claimants a whole year for exploration before having to make any payments at all, tends to encourage oil explorations and lifts the unnecessary limitations formerly placed on petroleum concessions. The purpose of this decree is to permit individuals to explore the subsoil in search of petroleum without the need of obtaining a concession or going through the formalities previously required by the law.